Preliminary Economic Assessment Technical Report for the East and West Zones Roughrider Uranium Project, Saskatchewan

Report Prepared for

Hathor Exploration Ltd.

Report Prepared by

srk consulting

SRK Consulting (Canada) Inc.
2CH012.000
Effective Date: September 13, 2011

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Hathor Exploration Ltd.

Suite 1810, 925 West Georgia Street
Vancouver, BC, V6C 3L2

e-mail: info@hathor.ca
website: www.hathor.ca
Tel: +1.604.684.6707
Fax: +1.604.684.9277

SRK Consulting (Canada) Inc.
Suite 2200 – 1066 West Hastings Street
Vancouver, BC V6E 3X2

e-mail: vancouver@srk.com
website: www.srk.com
Tel: +1.604.681.4196
Fax: +1.604.687.5532

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Authors:
Gordon Doerksen, P.Eng.
Bruce Fielder, P.Eng.
Iouri Iakovlev, P.Eng.
David Keller, P.Geo.
Mark Liskowich, P.Geo.
Bruce Murphy, FSAIMM
Cam Scott, P.Eng.

Peer Reviewed by:

Mark Liskowich, P.Geo

Executive Summary

This Preliminary Economic Assessment ("PEA") Technical Report was compiled by SRK Consulting (Canada) Inc. for Hathor Exploration Limited ("Hathor"). The PEA considers the economics of the East and West Zone mineral resources of the Roughrider Uranium Project. The Far East Zone, currently an exploration target and undergoing a mineral resource estimate, is not included in this study. Hathor owns 100% of the Roughrider Project.

Melis Engineering Ltd. ("Melis") of Saskatoon, Saskatchewan provided all metallurgy and mineral processing-related information for this study.

Property Description and Location

The Roughrider East and West Zones form part of the Roughrider Uranium Deposit within the Roughrider Project, previously referred to as the Midwest NorthEast Project. The project is located in the eastern Athabasca Basin of northern Saskatchewan, Canada. They are located approximately 7 km north of Points North Landing, a service centre at the terminus of Provincial Road 905, approximately 440 km north of La Ronge and approximately 700 km north of Saskatoon.

The Roughrider Project comprises three contiguous mineral leases (ML-5544, -5545, -5546) and covers an area of approximately 598 ha. The project area has an irregular shape with a north-south dimension of a maximum of 2.5 km along its eastern boundary and east-west dimension of a maximum of 3 km.

The mineral leases adjoin the north-eastern boundary of the Midwest Joint Venture's ("MWJV") (operated by Areva Resources Canada Inc.) mining lease, ML 5264, and are 4.7 km and 1.8 km northeast of Midwest and Midwest A uranium deposits of MWJV respectively. The Dawn Lake uranium deposits are also located approximately 6 km east of the Roughrider Project.

The core camp facility is located within the main lease, on the shore of the northeast bay of McMahon Lake and is located at 556,656 m East and 6,465,610 m North Universal Transverse Mercator NAD83 datum Zone 13 or Latitude 58.3275º North and Longitude 104.0325º West (WGS84 datum).

Under an agreement dated September 10, 2004, between Roughrider Uranium Corp. ("Roughrider") and Bullion Fund Inc. ("Bullion Fund"), Roughrider earned a 90% interest in claim S-107243 (and six other claims that became part of Roughrider's Russell South property) by paying Bullion Fund an aggregate of $200,000 cash. Bullion Fund retained a 10% carried interest. On August 10, 2006, Roughrider became a wholly owned subsidiary of Hathor. A two percent net smelter return on ML-5544 is payable to original Roughrider shareholders.

On April 12, 2007, Terra Ventures Inc. ("Terra") announced that it had closed a deal with Bullion Fund to acquire an 8% carried working interest in seven claims comprising 56,360 acres in two separate projects located in the Athabasca Basin, Saskatchewan, of which 90% of the remaining 92% working interest was held by Hathor. One of the claims was S-107243. Terra's interest is carried in all respects through to the completion of a feasibility study and the public announcement that the claims will be put into commercial production. Terra paid $2,300,000 to acquire the interest and also paid a finder's fee of $69,000.

On March 24, 2008, Terra announced that it had closed its agreement with Bullion Fund to purchase Bullion Fund's remaining 2% carried working interest in the Roughrider Project of Hathor. This purchase increased Terra's holding to a 10% carried working interest through to the completion of a feasibility study and the public announcement that the claims will be put into commercial production. The consideration paid by Terra to acquire this interest was $2,500,000 and 3,000,000 shares of Terra.

On April 18, 2011, Hathor and Terra announced that they had executed a binding letter agreement pursuant to which Hathor would acquire, in an all-share transaction, all of the issued and outstanding shares of Terra. On May 9, 2011, Hathor and Terra announced that they had executed a definitive plan of arrangement agreement (the "Arrangement") to complete the previously announced merger. The result of the arrangement is consolidation of 100% ownership of the Roughrider Deposit. On August 2, 2011, Terra received approval from 96% of votes cast at a special meeting of its shareholders held in Vancouver. On August 4, 2011, Terra received final approval from the Supreme Court of British Columbia to complete the Arrangement. On August 5, 2011, Hathor and Terra announced the completion of the plan of arrangement and Terra is now a wholly-owned subsidiary of Hathor.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

The property can be accessed either by helicopter, fixed wing aircraft or winter road from Points North Landing. Points North Landing is accessed from La Ronge via provincial Highway 102 to South End and then Provincial Road 905. The nearest, sizable population centre is La Ronge, approximately 440 km by road to the south. There is a daily commercial airline service from Saskatoon to Points North Landing.

The operating McClean Lake mill complex is located approximately 11 km (on a straight line distance) east of the Roughrider Project. Any mineralized material that might be extracted from the Roughrider Project could be transported the short distance to this facility for milling and treatment.

Electrical power is available from the provincial grid through a switching station at Points North Landing. Adequate water for a mining operation could be obtained from the lakes on the property. Uranium mining operations in the general area include the McArthur River, McClean Lake, and Eagle Point mines. Uranium milling facilities in the area are the McClean Lake mill and the Rabbit Lake mill

The climate is a mid-latitude continental climate, with temperatures ranging from 32ºC in the summer to -45ºC in the winter. Winters are long and cold, with mean monthly temperatures below freezing for seven months of the year. Annual precipitation is about 500 ml per year, with half of that in the summer months. Winter snow pack averages 70 cm centimetres to 90 cm. Lake ice forms by mid-October and usually melts by mid-April. Field operations are possible year round with the exception of limitations imposed by lakes and swamps and the periods of break-up and freeze-up.

The property is characterized by a relatively flat till plain with elevations ranging from 477 metres (South McMahon Lake) to 502 metres above mean sea level. Throughout the area, landforms distinctively trend northeast arising from passages of glacial ice from the northeast to the southwest or the property.

History

Between 1969 and 1974, following the discovery of the Rabbit Lake uranium deposit in 1968 by Gulf Minerals Ltd., Numac Oil and Gas held the large Permit Number Eight over the Midwest Lake and Dawn Lake areas. Prospecting, airborne radiometric surveys and lake sediment sampling for uranium and radon were carried out in 1969 and 1972. At the time, Numac Oil and Gas, in conjunction with their partners Esso Minerals and Bow Valley Industries, focused on the Midwest Lake area, located adjacent to Hathor's Roughrider Project.

In 1976, Asamera Oil Corporation initiated the Dawn Lake project, located approximately 6 km Southeast of the current Roughrider Project. Asamera discovered the Dawn Lake 11, 11A, 11B, and 14 zones in 1978.

In 1983, the Saskatchewan Mining and Development Corporation, predecessor to Cameco Corporation ("Cameco") became the operator of the Dawn Lake Joint Venture. By 1995, the Dawn Lake Joint Venture consisted of Cameco, Cogema Resources Inc., PNC Exploration Canada Ltd., and Kepco Canada Ltd. The Dawn Lake Joint Venture held the Esso North claim until it lapsed in 2003.

Early work by Asamera on the Esso North claim consisted of INPUT-electromagnetic and aeromagnetic surveys in 1977, followed by airborne very low frequency (VLF)-EM, magnetic and radiometric surveys in 1978 and 1979 by Kenting and Geoterrex, respectively.

These surveys located an east-west trending conductor of moderate strength and a radiometric anomaly associated with a broad VLF-EM response on the eastern portion of the Esso North claim. From 1978 to 1981, Turam, Vector Pulse EM, and VLF-EM surveys confirmed the east-west conductor as well as some weaker northeast trending VLF-EM conductors. The East-West conductor occurs just outside the western boundary of Hathor's lease ML-5544. During this same period, Asamera drilled 21 core bore holes on the Esso North claim. The first ten holes, EN-1 to EN-10, were drilled across the projected northeast strike extent of the Roughrider Project. These holes are located within Hathor's lease ML-5544 and penetrated basement rock for an average length of 25 m. The other eleven holes were drilled on the main east-west striking conductor. Results, however, were discouraging. Some evidence of structural disturbance and alteration was observed in sandstone intersected in holes EN-14, EN-15, and EN-16. Only EN-14 and EN-15 are collared within the current Hathor lease ML-5544.

In 1984, SMDC carried out Time Domain EM on the Esso North claim and completed two additional holes (Roy et al., 1984). Hole EN-18 targeted a weak TEM conductor in the vicinity of the east-west conductor. Results of this hole were negative. Drillhole EN-19 targeted a weak northeast trending TEM conductor, it intersected faulting and alteration in the Athabasca sandstone, but had no other interesting features. Holes EN-18 and EN-19 are located within Hathor's lease ML-5544.

Exploration on the Esso North claim was dormant until 1995, when Cameco resurveyed the area with TEM and located both the east-west conductor and the weak northeast striking conductor. The latter target was tested by one hole, EN-20; it intersected faulted and altered sandstone but had no significant radioactivity. Hole En-20 is located on Hathor lease ML-5544.

There are no historical mineral resource and/or mineral reserves estimates for the Roughrider East Zone. There has been no historical production from the Roughrider East Zone.

Geological Setting and Mineralization

The most significant uranium district in Canada is the Athabasca Basin which covers over 85,000 km^2 in northern Saskatchewan and north-eastern Alberta. The basin contains a relatively undeformed and unmetamorphosed sequence of Mesoproterozoic clastic rocks, the Athabasca Group. These rocks lie unconformably on the deformed and metamorphosed rocks of the Western Churchill Province of the Archean Canadian Shield. The basement rocks consist of Archean orthogneisses, which are overlain by, and structurally intercalated with, the highly deformed supracrustal Palaeoproterozoic Wollaston Group.

The Athabasca Basin is elongated along an east-west axis and straddles the boundary between two subdivisions of the Western Churchill Province; e the Rae Subprovince to the west and the Hearne Subprovince to the east. The subprovinces are separated by the northeast trending Snowbird Tectonic Zone, which is called the Virgin River-Black Lake shear zone in the area of the Athabasca Basin. In the vicinity of the Roughrider Project area within the eastern Athabasca Basin, the basement rocks of the Hearne Subprovince are subdivided into a number of domains. The Hearne Craton, beneath the eastern Athabasca Basin, comprises variably reworked Archean basement, which is dominated by granitic domes and foliated to gneissic granitoid rocks with infolded outliers of Paleoproterozoic metasedimentary rocks. The structural and tectonic regime of the area has been influenced strongly by collisional tectonics between the Hearne and Superior Cratons during the early Proterozoic Trans-Hudson Orogen, which occurred approximately 1.9 to 1.77 billion years (“Ga”) ago.

Unconformably overlying the basement rocks is the late Paleoproterozoic to Mesoproterozoic Athabasca Group consisting mainly of fluvial clastic sedimentary rocks. The Athabasca Group comprises eight formations in which four broadly fining-upward, unconformity-bound cycles can be distinguished. Overall, sedimentary provenance was from the east, south, and northwest. Lithologies are dominated by fine- to coarse-grained, partly pebbly or clay-intraclast-bearing quartz arenites and minor conglomerates.

Four important lithostructural domains have been identified in the Hearne Subprovince: the Eastern Wollaston Domain (“EWD”), Western Wollaston Domain (“WWD”), Wollaston-Mudjatik Transition Zone (“WMTZ”), and Mudjatik Domain (“MD”). The basement rocks within the property are part of the WMTZ.

The majority of the uranium occurrences and all currently producing uranium mines in the area are hosted in rocks of the WWD and WMTZ.

The Roughrider Uranium Deposit overlies the Wollaston-Mudjatik Transition Zone ("WMTZ") of the Wollaston Domain. The basement is structurally complex, comprising steeply dipping Wollaston Group rocks interfingering Archean granitic to granodioritic orthogneisses. Interpretations of aeromagnetic data suggest that several Archean granitic domes dominate the basement geology. Several basement packages are recognised in the Roughrider Uranium Deposit:

- Wollaston Group;
- Hanging Wall Wedge ("HWW");
- Foot Wall Wedge ("FWW"); and
- Midwest Dome ("MWD").

The property geology indicates that the Roughrider Project is situated in the basal part of the Wollaston Group, which is dominated by garnet- and cordierite-bearing pelitic gneisses with subordinate amounts of graphitic pelitic gneisses and psammopelitic to psammitic gneisses, and rare garnetites. Both the FWW and HWW are complex packages and comprise variable amounts of granitic to tonalitic orthogneiss that was subjected to local anatexis. The gneiss was intruded by younger pegmatites, leucogranites and microgranites. From drilling data, the FWW is only locally present and has been interpreted to plunge to the southwest and does not extend from the Roughrider West Zone eastward across the Roughrider East Zone. It is possible that several FWW are present, and these represent slivers of either the MWD or the FWW that had been carved off during deformation.

The MWD comprises strongly foliated orthogneisses that range in composition from granitic to dioritic. They additionally contain volumetrically minor amounts of partially melted material, and younger 'Hudsonian' pegmatites, leucogranites and micrograntes.

The sandstone and basement rocks have been subjected to several episodes of brittle deformation, including the brittle reactivation of older ductile shear zones.

Macro-scale geophysical, geological and structural modelling suggests that Roughrider Project is cross cut by a large number of structures. The two main structures to note are: (1) the east-west trending Roughrider Corridor and (2) the northern extension of the north northwest-south southwest- trending Midwest Trend, that hosts the Midwest and Midwest A Deposits on the adjacent mineral leases to the south of the project.

To date, core drilling on the Roughrider Project has focussed on mineralization at the Roughrider Uranium Deposit, which comprises three zones, Roughrider West, East and Far East Zones.

The Roughrider West Zone is centered on the Roughrider Structure, while the Roughrider East and Far East Zones are centred at or near the intersection of the Midwest Trend and Roughrider Corridor.

The Roughrider West Zone was discovered during the winter drilling program of February 2008. A hydrothermal clay alteration system was intersected in drillhole MWNE-08-10, while high-grade uranium mineralization (5.29% uranium oxide (“U_3O_8”) over a core length interval of 11.9 m) was intersected in drillhole MWNE-08-12. The Roughrider West Zone is defined by approximately 149 diamond drillholes, and has been intersected along a northeast-southwest strike length of approximately 200 m with an across-strike extent of 100 m. Uranium mineralization occurs at depths of 190 m to 290 m below surface and is hosted predominantly within basement rocks. Only minor amounts of uranium occur at or above the unconformity.

The Roughrider East Zone was discovered during the summer drilling program in September 2009. Hydrothermal alteration was intersected in a number of earlier drillholes during the summer program. High-grade uranium mineralization (12.71% U_3O_8 over a core length interval of twenty-eight metres) was intersected subsequently in drillhole MWNE-10-170. This zone was delineated by drilling during the winter and summer of 2010. The best intersection to date was obtained in drillhole MWNE-10-648, which intersected 7.75% U_3O_8 over a core length interval of 63.5 m. The Roughrider East Zone is currently defined by approximately 88 diamond drillholes (21 of which were used to evaluate the mineral resource), and has a surface projection of approximately 120 m long in a north-easterly direction, which corresponds to a down-dip length of approximately 125 m, and an across-strike extent of up to 70 m. Uranium mineralization has a vertical extent of up to eighty to 100 m, starting at depth approximately 250 m from surface, and some 30 m to 50 m below the unconformity. This is slightly deeper than the Roughrider West Zone. Mineralization forms moderately dipping, cigar-shaped shoots along the intersection of these two controlling structures.

A third zone, the Roughrider Far East Zone, was discovered during the winter drilling program in February 2011. The discovery drillhole intersected 1.57 % U_3O_8 over core length of 37.5 m. The current outline of the Far East Zone is defined by mineralization in 28 of 40 drillholes completed in the immediate vicinity of Roughrider Far East Zone; weak mineralization in other drillholes is not included in the current outline of the Far East Zone. The best intersection to date is drillhole MWNE-11-715, which intersected 7.91% U_3O_8 over a core length interval of 27.0 m.

As currently modelled, the West Zone occurs as a number of high grade lenses (greater than 3.0% U_3O_8 mantled by an envelope lower grade mineralization that is defined by a 0.05 % U_3O_8 cut off. The mineralization plunges moderately to the north or northwest. The contacts between these mineral lenses are sharp. In contrast, the East Zone comprises a series of stacked, parallel lenses of high grade material defined by 0.4% U_3O_8 that dip and plunge to the northeast. Unlike the West Zone, the lenses are not uniformly mantled by an extensive rim of low grade mineralization. Abundant low grade mineralization is present outside of the modeled mineral lenses in the current resource model, and represents additional resource potential.

Uranium mineralization is highly variable in thickness and style. High grade mineralization occurs primarily as medium- to coarse-grained, semi-massive to massive pitchblende with what has been termed worm-rock texture, and texturally complex redox-controlled mineralization. This high grade mineralization is intimately associated locally with lesser amounts of red to orange coloured oxy-hydroxillized iron oxides. Yellow secondary uranium minerals, probably uranophane, are present locally as veinlets or void-filling masses within the high grade primary mineralization.

Lower grade mineralization occurs as either disseminated grains of pitchblende, fracture-lining, or veins of pitchblende.

Galena occurs in a number of habits and is variably present in the uranium mineralization. The lead is presumed to have formed from the radioactive decay of uranium. Veinlets of galena are up to 5 mm thick and either crosscut massive pitchblende as anhedral masses (less than 1 mm in size) interstitial to the massive pitchblende, or as fine-grained, sub-millimetre-scale disseminated flecks of galena omnipresent throughout mineralized drill core. In all cases, the galena appears to have formed later than the uranium mineralization.

Mineralization is, in general terms, mono-metallic in composition. In the West Zone, visible, crystalline nickel-cobalt sulpharsenides are present locally in a number of drillholes (for example in drillholes, MWNE-08-12, MWNE-08-19, MWNE-08-37, MWNE-096-56, MWNE-09-90, MWNE-09-101, MWNE-10-188B, MWNE-10-210, MWNE-10-216). At the East Zone, the presence of nickel-cobalt sulpharsenides is rare. Only three individual sample assays returned values of greater than one percent arsenic, nickel or cobalt. Most samples contain less than 0.1% of these elements. The exact relationship of these elements to uranium is variable and still unclear at this time.

Unlike many unconformity-type uranium deposits in the Athabasca Basin, however, variable amounts of copper mineralization are present within the Roughrider Deposit.

Individual samples locally return values greater than 2% copper, up to a maximum of 17% copper. However, on a deposit scale, copper does not contribute significantly to the metal budget of the Roughrider Deposit.

Deposit Types

The target mineralization on the Roughrider Project is Athabasca unconformity-type uranium mineralization. Fundamental aspects of the Athabasca unconformity-type uranium deposit model include reactivated basement faults and two distinct hydrothermal fluids. Reactivation of brittle basement seated faults has fractured the overlying sandstones. The faults provide a plumbing system for reducing fluids that originate in the basement. These fluids interact with a second, oxidizing fluid that originates within the Athabasca sandstone stratigraphy. The latter fluid migrates through the inherent porosity in the sedimentary rocks. In appropriate circumstances, the two fluids mix and precipitate uranium in structural traps at or near the basal Athabasca-basement unconformity. Mineralization may occur (or may have been remobilized) in the controlling fault structures well above the unconformity.

Unconformity-associated uranium deposits are pods, veins, and semi-massive replacement bodies consisting mainly of pitchblende. These deposits typically occur close to basal unconformities, in particular those between Proterozoic conglomeratic sandstone basins and metamorphosed basement rocks. Prospective basins in Canada are filled by thin, relatively flat-lying, and widely unmetamorphosed but pervasively altered, Proterozoic (approximately 1.8 Ga to less than 1.55 Ga), mainly fluvial, red bed sediments. They comprise quartzose conglomerate, sandstone, and mudstone.

The basement gneiss was intensely weathered and deeply eroded with variably preserved thicknesses of reddened, clay-altered, hematitic regolith grading down through a green chloritic zone into fresh rock. The basement rocks typically comprise highly metamorphosed Archean to Paleoproterozoic granitoid and supracrustal gneiss including graphitic metapelites that hosts many of the uranium deposits. The bulk of the uranium-lead isochron ages on pitchblende are in the range of 1600 Ma to 1350 Ma.

Economic amounts of pitchblende mineralization typically occurs either as monometallic, generally basement-hosted pitchblende vein fill, breccia fillings, and replacements in fault zones or as polymetallic, commonly sub-horizontal, semi-massive replacement pitchblende lenses just above or straddling the unconformity. Polymetallic mineralization contains variable amounts of uranium, nickel, cobalt and arsenic as well as traces of gold, platinum-group elements, copper, rare-earth elements and iron.

Two end-members of the deposit model have been defined. A sandstone-hosted egress-type model (one example is the Midwest A deposit) involves the mixing of oxidizing sandstone-hosted brine with relatively reduced fluids from the basement in the sandstone. Basement-hosted, ingress-type deposits (one example is the Rabbit Lake deposit) formed by fluid-rock reactions between an oxidizing sandstone brine and the local wall rock of a basement fault zone. Both types of mineralization and associated host-rock alteration occurred at sites of basement–sandstone fluid interaction where a spatially stable redox gradient or front was present. Although either type of deposit can result in high grade pitchblende mineralization with up to 20% pitchblende, they are not physically large.

Unconformity-type uranium deposits are surrounded by extensive alteration envelopes. In the basement, these envelopes are typically narrow but become broader where they extend upwards into the Athabasca group for tens of metres to even 100 m or more above the unconformity. Hydrothermal alteration is variously marked by chloritization, tourmalinization, several episodes of hematization, illitization, silicification or desilicification, and dolomitization.

Exploration

Exploration work conducted at the Roughrider Project by Hathor includes a number of geophysical (electromagnetic, magnetic, gravity, seismic and resistivity) surveys completed by a number of different contractors between 2005 and 2009, and relogging of available historical drill core in 2006 by Hathor.

Fugro completed a 395 line kilometre airborne electromagnetic (TEMPEST) and magnetic gradiometer survey in 2007. The survey was aimed at identifying sandstone alteration features using early time electromagnetic channel data. Results showed a one kilometre wide region of early channel conductivity that coincided with a group of anomalies from ground resistivity surveys, including a low resistivity zone which is interpreted to identify the hydrothermal alteration associated with the Roughrider Uranium Deposit.

In 2007 and 2008, MWH Geo-Surveys Ltd. carried out several ground gravity surveys with particular emphasis on the immediate vicinity of the Roughrider Uranium Deposit. These surveys identified numerous gravity low features, interpreted to be due to alteration effects in sandstone.

In 2007, Hathor commissioned a detailed three dimensional (“3D”) seismic reflection survey at the Roughrider Project. A first pass qualitative interpretation of the seismic data was achieved using Gocad’s 3D imaging software. 3D analysis of the data generated a framework of prominent features, mostly oriented at 65º, which were interpreted to represent major structural components within the survey area. The interpretation will be updated and refined with the addition of results, including faults and alteration zones, as encountered in subsequent exploration drilling.

Between 2007 and 2009, six resistivity surveys were completed at the Roughrider Project, with a combined total of 125.05 km of profile coverage. Modelling results (using the Loki code) indicate intense shallow, possibly surficial, low resistivity material in the north western region of McMahon Lake. At mid-sandstone depths (74 m to 136 m) an east-west trending corridor of low resistivity has been interpreted to represent alteration features in the sandstone associated with the underlying mineralization. This interpretation includes the known mineralization of the Roughrider Uranium Deposit.

Geotech Ltd. carried out a 568 line kilometre helicopter-borne time-domain electromagnetic (“VTEM”) survey at the Roughrider Project in 2008. Principal geophysical sensors included a VTEM system and a caesium magnetometer. Ancillary equipment included a GPS navigation system and a radar altimeter. A total of 568 line kilometres were flown. This VTEM survey was designed to locate bedrock conductors on the Roughrider Project, but none were found

Drilling

Hathor has conducted a number of drilling programs on the Roughrider Project since 2007.

A summary of drilling completed to date since 2007 comprises 389 holes or 143,414 m of diamond drilling over the entire Roughrider Project. All drilling since Summer 2008 has been contracted to TEAM Drilling LP, from Saskatoon. These programs have utilized ZMC A5 drills, with a depth capacity in excess of 600 metres with BQ, NQ and HQ-sized rods.

Winter drill programs utilize drills mounted on metal skids to allow mobilization between drill collar sites. Summer drill programs utilized a combination of skid-mounted and helicopter-portable drill rigs. These rigs can complete drillholes ranging in dip from vertical to 45º. NQ-sized holes were cased NW into bedrock and drilled NQ size to depth. In rare instances NQ-sized holes were reduced to BQ-sized holes due to severely bad ground.

All mineralized and non-mineralized holes within the Roughrider West, East and Far East Zones were cemented from bottom to top. All land-based drill whole locations are marked with a tagged picket as per Saskatchewan Ministry of Environment regulations.

Diamond drilling for the Roughrider East and West Zones has been undertaken using a grid pattern with intersections for the Roughrider West Zone at approximately 10 m by 10 m and 20 m to 15 m intervals for the Roughrider East Zone. .

Drillholes are both vertical (inclination of 85º to 90º down) and inclined down to a maximum of 45º. Inclined drillholes are orientated at an azimuth of approximately 090º, 135º, 160º, and 340º.

Holes are spotted on a grid and collar sites are surveyed by differential GPS using NAD83 and UTM Zone 13. Down hole surveys were completed either with, or a combination of, Reflex EZ-Shot or a Reflex Gyro instrument.

Where possible, at the completion of each drillhole, downhole radiometric surveys were performed down the drill string at a speed of 15 m per second down hole and 5 m per second up hole. All holes were surveyed using a Mount Sopris winch and matrix logger interface board.

Unmineralized or weakly mineralized holes were surveyed using a single crystal (NaI) gamma probe. Holes with an estimated pitchblende content greater than three percent were surveyed with a down hole triple (one NaI and two Geiger-Mueller tubes) gamma probe. The Saskatchewan Research Council (“SRC”) provides down hole calibration test pit facilities in Saskatoon, Saskatchewan, for the calibration of down hole gamma probes. These test pits consist of four variably mineralized holes with maximum grades of 0.61%, 0.30%, 1.35%, 4.15% pitchblende.

The probes used for the surveys were calibrated at the SRC test pit facility and allow for grade thickness estimates to be made from the instrument readings and grade estimates equivalent to U_3O_8 (“*e* U_3O_8”) to be calculated. However, it must be noted that, in general, no calibrations are available for high-grade mineralization (more than five percent U_3O_8) because Hathor has not yet been able to maintain an open, cased hole in such material and the highest grade SRC test pit available is 4.15% U_3O_8. Consequently, no *e* U_3O_8 grades are generally reported.

Sample Preparation, Analyses and Security

Drill core from the Roughrider Project was logged, marked for sampling, split, bagged, and sealed for shipment by Hathor personnel at their fenced core-logging facility on the property. All samples for pitchblende assay were transported by land, in compliance with pertinent federal and provincial regulations by Hathor personnel. The sample containers were transported directly to the Geoanalytical Laboratories of the SRC located in Saskatoon.

The Geoanalytical Laboratories of the SRC are unique facilities offering high quality analytical services to the exploration industry. The laboratory is accredited ISO/IEC 17025 by the Standards Council of Canada for certain testing procedures including those used to assay samples submitted by Hathor.

The laboratory is licensed by the Canadian Nuclear Safety Commission ("CNSC") for possession, transfer, import, export, use and storage of designated nuclear substances by CNSC Licence Number 01784-1-09.3. As such, the laboratory is closely monitored and inspected by the CNSC for compliance.

SRC is an independent laboratory, and no associate, employee, officer or director of Hathor is, or ever has been involved in any aspect of sample preparation or analysis on samples from the Roughrider East Zone.

SRC performs the following sample preparation procedures on all samples submitted to them. There is no sample preparation involved for the samples sent for clay analyses.

On arrival at SRC, samples were sorted into their matrix types (sandstone or basement rock) and according to radioactivity level. The samples were prepared and analyzed in that order.

Sample preparation (drying, crushing, and grinding) was done in separate facilities for sandstone and basement samples to reduce the probability of sample cross-contamination. Crushing and grinding of radioactive samples yielding more than 2,000 cps was done in another separate CNSC-licensed radioactive sample preparation facility. Radioactive material was kept in a CNSC-licensed concrete bunker until it could be transported by certified employees to the radioactive sample preparation facility.

Sample drying was carried out at 80ºC with the samples in their original bags in large low temperature ovens. Following drying, the samples were crushed to 60% passing 2 ml using a steel jaw crusher. A 100 g to 200 g split was taken of the crushed material using a riffle splitter. This split was then ground to ninety percent passing 150 mesh using a chromium-steel puck-and-ring grinding mill for mineralized samples or a motorized agate mortar and pestle grinding mill for all non-mineralized samples.

The resulting pulp was transferred to a clear plastic snap-top vial with the sample number labelled on the top. All grinding mills were cleaned between sample runs using steel wool and compressed air. Between-sample grinds of silica sand were performed in case the samples were clay-rich.

All split core samples, both mineralized and non-mineralized, from within the mineralized section were assayed for U_3O_8using SRC accredited U_3O_8-method.

Hathor relied partly on the internal analytical quality control measures implemented by SRC. In addition, Hathor implemented external analytical quality control measures consisting of using control samples in all sample batches submitted for assaying and requesting replicate pulp assays for every 20th sample. External quality control checks consisted of inserting a field duplicate approximately every ten metres of sampling and a field blank approximately every 10 m.

SRC used five control samples including five standards prepared by CANMET of Natural Resources Canada.

Umpire assays were sent to SRC Analytical Laboratory and analysed using Delayed Neutron Counting for uranium analysis.

From January 2009 (starting from drillhole MWNE 09-43A onwards), Hathor measured dry specific gravity on un-split core samples from various host rocks and mineralization styles. Specific gravity for whole core samples was measured at the exploration project site using a weight in water weight in air method. Samples are dried for a four days. Dry specific gravity was determined by the water immersion methodology. Dried core pieces were weighed, wrapped in plastic film which was heated to make tight seal around the core, and then weighed suspended in water. Dry specific gravity was determined for 50 cm core lengths to correspond to the sample interval. Prior to the determination of specific gravity of the unknown samples, dry specific gravity is determined on three in-house standards.

For quality control purposes, 26 and six core specific gravity samples from the Roughrider West and East Zones, respectively were submitted to an umpire laboratory at SRC.

The specific gravity database used in East Zone resource estimation consists of 89 dry specific gravity determinations, including 46 measurements on core samples from the mineralization envelope. In contrast, the specific gravity database used in the West Zone resource estimation consists of 366 dry specific gravity determinations from the mineralized envelope. In the opinion of SRK the sampling preparation, security and analytical procedures used by Hathor are consistent with generally accepted industry best practices and are therefore adequate.

Data Verifications

Hathor relied partly on the internal analytical quality control measures implemented by SRC and also implemented external analytical quality control measures consisting of control samples in all sample batches submitted for assaying.

During drilling, experienced Hathor geologists implement practical measures designed to ensure the reliability and trustworthiness of exploration data acquired on the Roughrider Project. In the opinion of SRK, the field procedures used by Hathor generally meets or exceeds “industry best practices”. Sample shipments and assay deliveries were routinely monitored as produced by the primary laboratory.

In accordance with the National Instrument 43-101 guidelines, G. David Keller from SRK visited the Roughrider Project during the period of September 13 to 14, 2010 and March 16 to 18, 2011. During the time of the site visit drilling operations were ongoing. The purpose of the site visit was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property and assess logistical aspects and other constraints relating to conducting exploration work in this area.

All aspects that could materially impact the mineral resource evaluation reported herein were reviewed with Hathor staff. SRK was given full access to all relevant project data. SRK was able to interview exploration staff to ascertain exploration procedures and protocols.

Drillhole collars are clearly marked with stakes inscribed with the borehole number on an aluminum Dynamo labels. No discrepancies were found between the location, numbering or orientation of the holes verified in the field and on plans and the database examined by SRK.

During the site visit SRK examined and briefly relogged mineralized zones for 18 boreholes within the West zone, 16 drillholes with the East Zone and five holes within the Far East Zone.

During drilling, experienced Hathor geologists implement practical measures designed to ensure the reliability and trustworthiness of exploration data acquired on the Roughrider Project. In the opinion of SRK, the field procedures used by Hathor generally meet or exceed “industry best practices”.

In the opinion of SRK, the assay analytical results delivered by SRC are sufficiently reliable for the purpose of resource estimation.

Adjacent Properties

An adjacent property to the Roughrider Project is the Waterbury Lake Project, consisting 13 mineral dispositions (40,256 hectares) held by the Waterbury Lake Uranium Corporation, a Limited Partnership owned by Fission Energy Corp. (50%) and Korea Waterbury Uranium Corp. (50%). Disposition S-107370 hosts the J- and J-East Zones. No NI 43-101 technical reports are available for the J - or J - East Zones.

Uranium mineralization occurs primarily at the unconformity, but may extend as a keel into the basement, immediately below the unconformity (McElroy 2010). Drill intersections in the J-Zone are characterized by a broad continuous zone of alteration extending from several metres above the unconformity to greater than 25.0 m below the unconformity, with uranium occurring within this altered system.

J - East Zone, located approximately 15.0 m to west of the claim boundary is currently defined by three boreholes. The best intersection to date is Hole WAT10-102, which intersected a core length interval of 8.50 m grading 0.38% U_3O_8 (220.00 -228.50 metres) and 3.5 m of 0.67% U_3O_8 (233.00-236.50 m) (source: Fission Energy Corp. website). The uranium mineralization is hosted in basement rock and is similar to that found at the Roughrider Uranium Deposit. The J-East Zone may represent an extension of the Roughrider Uranium Deposit.

Metallurgical Testing

Metallurgical testing has not been completed on representative samples from the Roughrider East Zone. Testing was, however, conducted on representative uranium mineralization samples from the nearby Roughrider West Zone, the characteristics of which are similar to Roughrider East Zone. SRK therefore believes that it is relevant to present the testing results obtained from Roughrider West Zone. Metallurgical testing on the Roughrider East Zone is planned for later in 2011 or 2012.

Mineral Processing

The milling process used by Melis Engineering Ltd. in the preparation of mill capital and operating cost estimates for Hathor’s Roughrider uranium project was a grind/acid leach/resin-in-pulp process, a description of which is summarized below. This process was used as the initial basis of the scoping study estimates. It remains to be further assessed with further metallurgical testing and trade-off studies as mining plans are developed and as the project advances.

Run-of-mine ore, at an average grade of 3.3% U_3O_8, will be received as high grade and low grade mineralization and stockpiled close to the mill for feeding a blend into a dump hopper and jaw crusher. Crushed ore will be discharged to a coarse ore bin to provide some surge capacity ahead of a SAG (semi-autogenous grinding) mill and ball mill grinding circuit. Ground slurry, at a product K_{80} size (80% passing size) of 125 microns, will be thickened prior to feeding the leach circuit.

Leaching will be completed in rubber lined steel tanks using sulphuric acid and sodium chlorate oxidant. A free acid level of 15 g H_2SO_4/L, an oxidation/reduction potential of 475 mV maintained with 4 kg $NaClO_3$/t, a 50 °C leach temperature and 12 hours retention time will provide a leach uranium extraction efficiency of 98.5%. Accounting for soluble and insoluble losses the overall uranium recovery is estimated at 97.7%.

The soluble uranium in the leach slurry will be recovered in an eight stage resin-in-pulp ion exchange circuit with uranium selective resin advancing counter-currently to the leach slurry flow. The loaded resin will be eluted with sulphuric acid to produce a concentrated liquor of the recovered uranium. Resin technologies for uranium recovery have advanced sufficiently in recent years to make this type of circuit a viable option to a CCD (counter current decantation)/solvent extraction circuit and overall capital have been shown to be lower.

The concentrated eluate from the resin-in-pulp circuit will be partially neutralized for gypsum removal ahead of uranium (yellowcake) precipitation with hydrogen peroxide and magnesium oxide for pH control. The yellowcake product will be calcined prior to packaging in 400 kg drums for shipment to a uranium refinery.

Leach residue and waste solution streams will be bulk neutralized with lime and treated with barium chloride and ferric sulphate prior to discharge to the tailings management facility. Tailings decant water and excess minewater will treated in a four stage treatment circuit, a first stage reverse osmosis (“RO”) circuit followed by three stage treatment of the RO reject using lime, barium chloride and ferric sulphate. RO permeate will be recycled to the mill for use as a process water. The treated effluent will pass through sand filters to remove residual suspended solids prior to discharge to the environment.

Mineral Resource Estimates

Hathor commissioned SRK to prepare an updated mineral resource estimate for the Roughrider Uranium Deposit on September 7, 2010. This represents the second mineral resource evaluation prepared for this deposit. A previous mineral resource statement was prepared in 2009 by Scott Wilson RPA Inc., as documented in a technical report dated December 18, 2009.

The resource estimation work for the West Zone was completed by G. David Keller, P. Geo (APGO#1235) with the assistance of Dominic Chartier, P. Geo (APQ#874), both "independent qualified persons" for the purpose of National Instrument 43-101. The effective date of this resource statement was November 29, 2010.

The resource estimation work was completed for the Roughrider East Zone by G. David Keller, P.Geo. (APGO#1235) and Sébastien Bernier, P.Geo. (APGO#1847), both appropriate "independent qualified persons" as this term is defined in National Instrument 43-101. The effective date of this resource statement was May 17, 2011.

The boundaries for uranium mineralization were initially modelled using LeapFrog software to produce U_3O_8 grade shells using a 0.05% U_3O_8 threshold for West Zone low grade domain, 3.0% U_3O_8 threshold for the high grade domain.

For the East Zone, only high grade domains were modelled using a U_3O_8 threshold of 0.5%. Using these grade shells Hathor developed wireframe solids for both Zones. For the West Zone 11 sub-zones in a high grade domain were model as well as four sub-zones in a low grade domain. All high grade sub-zones were contained in the largest low grade sub-zone. For the East zone seven stacked high grade zones were modelled. SRK reviewed the wireframe models developed by Hathor and carried out minor revisions making sure that all wireframes were snapped to drillhole contacts

Specific gravity varies significantly in both the Roughrider West and East Zones and therefore has to be estimated for each of the sub-zones.

Both West and East Zone assay and specific gravity data were composited at 0.5 m to provide common support for statistical analysis, variography and resource estimation. Outlier analysis on assay and specific gravity data indicated that capping was not required for West and East zone composites.

Traditional and normal scores variograms were used to model the spatial distribution of U_3O_8 and specific gravity for the West Zone. Variogram models were developed for one high grade sub-zone and one low grade sub zone. Other variogram models were developed using combined sub-zones for each of the high and low grade domains. For the East Zone a single variogram model for U_3O_8 composites was developed. Specific gravity data was too limited to develop variograms.

For the West Zone, U_3O_8 grades were estimated using three estimation runs. The first estimation run is based on a search ellipse with ranges equal to the largest variogram model structure the second run consists of a search ellipse range equal to twice the variogram range. The third estimation run consist of a search ellipse generally three times the search ellipse range. The bulk of blocks are estimated in the first run. Specific gravity was estimated using the same parameters.

For domains with limited specific gravity data, global kriging values were substituted for unestimated blocks in each of three sub-zones. Global kriging values of the entire high or low grade data sets were used for three sub-zones without density data. Potentially deleterious elements (arsenic, cobalt, copper, molybdenum, nickel and selenium) were estimated using ordinary kriging. The same estimation parameters as U_3O_8 were used for estimating these elements.

For the East Zone U_3O_8 grades were estimated using three ordinary kriging estimation runs using composite data from each domain, separately. The first estimation run considers a search ellipse with ranges equal to the largest variogram model structure. The second run considers a search ellipse equal to twice the variogram ranges. For a third estimation run the search ellipse was inflated to four times the variogram ranges. The bulk of blocks are estimated by the first run.

Estimation of specific gravity using composites provides the most reasonable means for defining specific gravity for this Zone. Specific gravity was estimated using an inverse distance squared function. Three sub-zones with very limited specific gravity data were assigned an average of specific gravity based on each sub-zone dataset. One sub-zone without specific gravity data was assigned an average of all data.

Potentially deleterious elements (arsenic, cobalt, copper, molybdenum, nickel and selenium) were estimated using ordinary kriging. Variogram models for U_3O_8 were assumed for these metals. The same estimation parameters as U_3O_8 were used for estimating these elements.

Mineral Resource Statement

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The mineral resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

The Mineral Resource Statement presented in Table 1 was prepared by G. David Keller, P. Geo. (APGO#1235) for the West Zone and G. David Keller, P.Geo. (APGO#1235) and Sébastien Bernier P.Geo. (APGO#1847) for the East Zone, all "independent qualified persons" as this term is defined in National Instrument 43-101. The effective data for the West Zone Mineral Resource Statement is November 29, 2010 and the East Zone Mineral Resource Statement is May 6, 2011.

Table 1: Mineral Resource Statement* for the Roughrider Uranium Deposit, Saskatchewan, SRK Consulting Inc., November 29, 2010 (West Zone) and, May 6, 2011 (East Zone)

Category	Quantity	Grade							Contained
	[Tonnes]	U_3O_8 [%]	As [%]	Co [%]	Cu [%]	Mo [%]	Ni [%]	Se [ppm]	U_3O_8 [lb]
West Zone									
Indicated High Grade Zone	58,200	10.68	0.17	0.03	0.41	0.22	0.15	46	13,703,000
Inferred High Grade Zone	36,600	13.07	0.69	0.10	0.57	0.26	0.55	56	10,546,000
Indicated Low Grade Zone	336,000	0.48	0.00	0.00	0.00	0.00	0.00	8	3,556,000
Inferred Low Grade Zone	7,000	0.31	0.00	0.00	0.00	0.00	0.00	4	48,000
Total Indicated West Zone	**394,200**	**1.98**	**0.03**	**0.00**	**0.06**	**0.03**	**0.02**	**13**	**17,207,000**
Total Inferred West Zone	**43,600**	**11.03**	**0.58**	**0.08**	**0.48**	**0.22**	**0.47**	**48**	**10,602,000**
East Zone									
Inferred Zone 1	26,000	12.17	0.02	0.01	1.49	0.05	0.01	15	6,970,000
Inferred Zone 2	30,000	13.34	0.03	0.01	1.34	0.13	0.02	49	8,930,000
Inferred Zone 3	32,000	17.39	0.03	0.01	0.15	0.14	0.02	22	12,140,000
Inferred Zone 4	3,000	1.34	0.00	0.00	0.08	0.03	0.00	9	80,000
Inferred Zone 5	11,000	1.65	0.01	0.01	0.40	0.14	0.01	14	390,000
Inferred Zone 6	12,000	3.57	0.01	0.01	1.05	0.06	0.01	31	940,000
Inferred Zone 7	5,000	6.84	0.00	0.00	0.04	0.03	0.00	13	680,000
Total East Zone Inferred	**118,000**	**11.58**	**0.02**	**0.01**	**0.86**	**0.10**	**0.02**	**27**	**30,130,000**
Combined East and West Zones									
Total Indicated	**394,200**	**1.98**	**0.03**	**0.00**	**0.06**	**0.03**	**0.02**	**13**	**17,207,000**
Total Inferred	**161,600**	**11.43**	**0.17**	**0.03**	**0.76**	**0.14**	**0.14**	**32**	**40,730,000**

*CIM Definition Standards have been followed for classification of mineral resources. The cut-off grade of 0.05 percent U_3O_8 for the West Zone and 0.4 percent U3O8 was for the East zone. U_3O_8 price of US$80 per pound and assumed. Reasonable prospect for economic extraction" assumes open pit extraction for West Zone and underground extraction for East Zone and metallurgical recovery of 98 percent. Mineral resources are not mineral reserves and do not have demonstrated economic viability. Totals may not add correctly due to rounding.

Regulatory Setting

The environmental assessment and regulatory setting in Saskatchewan is a two tiered system composed of an environmental assessment (EA) and a permitting/licensing component. Potential mining developments in the province must undergo an initial screening to determine the level of federal and/or provincial assessment the proposed project will require. This screening process, as well as any subsequent environmental assessment, is conducted in accordance with the Canadian Environmental Assessment Act and Saskatchewan's Environmental Assessment Act. Following a successful EA the project would be required to undergo an extensive construction application phase followed by an operating licensing/permitting phase. Subsequent to receipt of approvals to operate, the project would be regulated throughout its full life cycle (construction, operation, decommissioning and reclamation, institutional control) by federal and provincial departments and agencies, including but not limited to, the Canadian Nuclear Safety Commission (CNSC), Environment Canada (EC), the department of Fisheries and Oceans Canada (DFO) and Saskatchewan's Ministry of Environment (MOE).

The Roughrider project, as it is currently defined, will be required to undergo a full provincial Environmental Impact Assessment and a Comprehensive Study Assessment under federal legislation. In accordance with the Saskatchewan/Canada cooperation agreement, these separate EAs would be harmonized under a single administrative process that would be administered jointly by Saskatchewan's Environmental Assessment Branch and the Canadian Environmental Assessment Agency.

It is anticipated the harmonized environmental assessment of the proposed Roughrider Project will require between 36 and 48 months to complete with an associated cost of approximately $5 million. In recent years, the Federal Government has established the Major Projects Office Management in an attempt to reduce the length of the assessment period.

Environmental Considerations

The most significant environmental considerations associated with the Roughrider project are centered on the management of the waste streams associated with an underground uranium mine and milling project. With the primary focus being placed on the potential impacts associated with the management of tailings, mill effluent and mine water and the proponent's commitment to effectively mitigate the potential impacts associated with these waste streams, throughout operations, decommissioning and reclamation.

Engineering and environmental investigations required for final characterization of the potential impacts and to subsequently support the final design of the waste management facilities have not yet been completed. However, the current level of engineering associated with these facilities reflects industry best practices.

As such, it is reasonable to assume the waste management practises proposed in this study will not be considered fatal flaws during the proposed project's environmental assessment provided that the appropriate level of engineering and environmental investigations are completed as part of the project's next level of engineering.

A conceptual decommissioning and reclamation plan for the proposed tailings management facility has been developed along with a cost estimate. This closure plan has been designed to:

- Restrict or eliminate the migration of all potential contaminants of concern from all sources on the mine site;
- Restrict or eliminate all potential radiological exposures to animal or humans;
- Restrict or eliminate all potential public safety risks associated with the decommissioned and reclaimed mine site; and
- Return the property, to the extent possible, to pre-mining conditions

The conceptual decommissioning and reclamation plan is estimated to cost approximately $14.2 million.

Social Considerations

The uranium mining industry and the Government of Saskatchewan have focused significant effort towards obtaining a "social license" to operate in the Athabasca Basin region over the course of the past 20+ years. To this end many committees, working groups, partnerships and agreements have been formed between the uranium mining companies and the First Nations, Metis and non-First Nation communities. An open and transparent dialogue with the appropriate stakeholder groups is a fundamental step towards receiving a "social license" as well as a mandatory component to an environmental assessment.

Hathor has identified the key stakeholder groups that will desire involvement with the continued development of their project and they have initiated preliminary public consultations with representatives of these groups. Hathor recognizes the fundamental areas of concern, from an environmental protection and socio-economic perspective, expressed by these stakeholder groups and is committed to working with these groups to address these concerns.

Tailings and Waste Rock Management

Tailings will be stored in a purpose-built excavation situated on a gently sloping side hill approximately 1km northeast of the proposed plant site. The principal design objectives for the tailings storage facility ("TSF") require that all tailings be kept below the local water table and that there be a positive gradient into the TSF from the surrounding natural ground. The latter objective will be met by using the pervious surround concept which is based on placement of permeable, granular material, plus a filter layer, beneath and around the entire TSF perimeter. Pumps will be installed in the granular material to keep the water level in this material below the local groundwater table.

The storage quantity has been based on the latest estimate of mill throughput and a settled dry density of 0.9 tonnes per cubic metre. An allowance for an ice entrainment has also been included. Based on these criteria, the excavation will be approximately 750 m long by 200 m wide and approximately 30 m deep. The tailings will be a layer approximately 15 m overlying the basal granular and filter layers. At closure, the tailings will be capped with granular material, a filter layer and a layer of material suitable for sustaining vegetation

Mine Plan

The Roughrider deposits contain high-grade (>10% U_3O_8) mineralization and therefore a raisebore ("RB") mining method was selected to allow for remote mining operations and blending of high-grade and low grade or waste zones to produce a mill feed of 3.3% U_3O_8.

It was assumed that the sandstones and unconformity, lying about 200 m below surface, are water bearing and can only be developed with grout or freeze cover. Although most of the mineralized zones are located in basement granites, they are close enough to the unconformity (0-30 m) that the impact of any potential water ingress from the overlying sandstones and unconformity is planned to be mitigated by establishing a freeze-wall that umbrellas the deposits. Ventilation raises will also be mined inside freeze walls.

The main access into the mine is planned to be provided via a decline developed under a grout cover. The decline is planned to be developed in the sandstones with the best geotechnical and hydrogeological characteristics. The ramp is planned to pass through the unconformity in an area of good rock quality based on drilling results.

The mine design for the West and East deposits was initiated with the development of input parameters to estimate the cut-off grade. These parameters included estimates of metal price (US$60/lb U_3O_8), exchange rate, processing and mining costs, mining dilution, mill recovery, and royalties.

UG mine planning used the preliminary input parameters as shown in Table 2 to provide initial mineable shapes.

Table 2: Underground Preliminary Planning Parameters

Item	Unit	Value
Metal Recovery		
U_3O_8 Price	$US/lb U_3O_8	60
Exchange Rate	$C/$US	1.05
U_3O_8 Price	$C/lb U_3O_8	63.16
Payable Metal	% U_3O_8	100
Process Recovery	%	96*
Refining/Freight/Insurance/ Marketing	$C/lb U_3O_8	N/A
Royalties @ 5% NSR	$C/lb U_3O_8	3.08*
Net U_3O_8 price	$C/lb U_3O_8	58.50
OPEX Estimates		
Mining Cost	$ /t milled	425*
Processing Cost	$ /t milled	410*
G&A/Sustaining capital cost	$ /t milled	126*
Total Site Cost	$ /t milled	961*
Cut-off Grade		
Plant feed Cut-off Grade	% U_3O_8	0.75
Dilution	%	30
In-situ Cut-off Grade	% U_3O_8	1.06

**Preliminary estimates only. These were later changed as costs and recoveries were refined.*

The mineral resources contained in the proposed life of mine (“LOM”) plan are shown in Table 3. The East zone contains only inferred resources while the West zone contains a combination of indicated and inferred resources. The annual uranium production under the design criteria is estimated to be 5.0 Mlb U_3O_8 for a little more than 10 years of operations.

Table 3: Roughrider LOM Resource

Zone	Classification	Diluted Tonnes (t)	Cut-off Grade (U_3O_8%)	Diluted Grade (U_3O_8%)	Contained Metal (Mlb U_3O_8)
West	Indicated	286,000	1.06	2.5	15.8
	Inferred	58,000	1.06	7.2	9.2
East	Inferred	388,000	1.06	3.3	28.2

Capital and Operating Cost Estimates

Capital ("CAPEX") and operating ("OPEX") cost estimates were based on 2011 prices and are a combination of first principle calculations, factored costs for similar projects, vendor quotes and estimates based on experience. Tables 4 and 5 show summaries of the OPEX and CAPEX cost estimates.

Table 4: Unit OPEX Estimate Summary

Operating Costs	Unit	Unit OPEX Estimate
Mining	$/t milled	421
Processing	$/t milled	480
General and Administration	$/t milled	126
Average Unit operating Cost	**$/t milled**	**1,026**
Average Unit operating Cost	**$/lb U_3O_8**	**14.4**

Table 5: Capital Cost Estimate Summary

Item	Unit ($)	Total	Pre-production	Sustaining
Underground Mine	M$	**159**	95	64
Processing Facility	M$	**172**	172	0
Site Infrastructure	M$	**53**	51	2
Owner's Costs	M$	**8**	8	0
Closure	M$	**14**	0	14
EPCM (15%)	M$	**48**	48	0
Contingency (25%)	M$	**114**	94	20
Total Capital Cost	**M$**	**567**	**468**	**100**

Economic Analysis

An engineering economic analysis for the project was developed using earnings before interest and taxes. The starting date for the analysis was selected to be at the production decision point for the project, post feasibility study and EA, and therefore does not include costs to further drill off the deposit conduct engineering studies and conduct an EA and obtain permits. These pre-production-decision costs would likely be in the $20M to $40M range.

Three cases were run to provide a range of U_3O_8 prices and estimate their impact on the economic results. Cases A, B and C used U_3O_8 prices of US$60/lb. $70/lb and $90/lb respectively. The economic analysis shows that the project is very robust for all cases as summarized in Table 6.

An exchange rate of C$1.05:US$ was assumed for each case.

The break-even U_3O_8 price for the project is estimated to be US$31/lb U_3O_8.

Table 6: Economic Analysis Results

Parameter	Unit	Case A	Case B	Case C
U_3O_8 Price	US$/lb U_3O_8	60	70	90
Royalty Payments	M$	366	463	669
Pre-tax $NPV_{0\%}$	M$	1,592	2,042	2,928
Pre-tax $NPV_{7\%}$	M$	769	1,025	1,527
Pre-tax IRR	%	32	38	48
Pre-tax payback period	Production years	2.2	1.8	1.4

Economic Sensitivities

Sensitivity analyses were conducted for all three cases by individually modifying the capital cost, operating cost, metal price and grade up and down by 20% to show the sensitivity of the pre-tax $NPV_{7\%}$. The results show that the project is very robust. Like most mining projects, the project is most sensitive to commodity price and mill feed grade. For Case A, a 20% increase in U_3O_8 price leads to a 40% increase in pre-tax $NPV_{7\%}$ from $769M to $1,076M. A change in grade by 20% has a similar effect on pre-tax $NPV_{7\%}$. The converse occurs if the metal price drops by 20%, the pre-tax $NPV_{7\%}$ drops from $769M to $443M.

Operating costs and capital costs were approximately equal in terms of sensitivity. A 20% increase in operating costs in Case A reduces the pre-tax $NPV_{7\%}$ by $86M from $769M to $683M or -11%.

A summary of the sensitivity analysis is shown in Table 7.

Table 7: Sensitivity Analysis Results

Case	Variable	Pre-tax $NPV_{7\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A US$60/lb U_3O_8	Capital Cost	862	769	676
	Operating Cost	856	769	683
	Metal Price	443	769	1,076
	Grade	423	769	1,096
Case B US$70/lb U_3O_8	Capital Cost	1,118	1,025	932
	Operating Cost	1,111	1,025	939
	Metal Price	648	1,025	1,377
	Grade	647	1,025	1,400
Case C US$90/lb U_3O_8	Capital Cost	1,620	1,527	1,434
	Operating Cost	1,613	1,527	1,441
	Metal Price	1,076	1,527	1,978
	Grade	1,053	1,527	2,001

Conclusions

Exploration work by Hathor is professionally managed and uses procedures meeting or exceeding generally accepted industry best practices. After review, SRK is of the opinion that the exploration data collected by Hathor are sufficiently reliable to interpret with confidence the boundaries of the uranium mineralization for the Roughrider East and West Zones and support evaluation and classification of mineral resources in accordance with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" and CIM "Definition Standards for Mineral Resources and Mineral Reserves" guidelines.

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Roughrider deposit. There is adequate geological and other pertinent data available to generate a PEA. No fatal flaws were discovered during the study.

Based on current knowledge and assumptions, the results of this study show that the project is economic and should be advanced to the next level of study.

Risks

While there are many risks associated with most early-stage mining projects, many of those risks can be mitigated with appropriate information gathering and engineering work. The main risks associated with the Roughrider Project are, in summary:

- Geological Interpretation;
- Mineral Resource Classification;
- U_3O_8 price and exchange rate;
- The ability to secure environmental permits;
- The ability to achieve operating and capital cost estimates;
- Mine plan and mining method applicability;
- The ability to understand and manage ground water and ground conditions in the mine; and
- The ability to meet dilution and extraction expectations.

Opportunities

The following opportunities may improve the project economics:

- Potential increase in mineral resources by evaluating uranium mineralization located in the Far East Zone (adjacent to the Roughrider East Zone); and

  Potential up-grade of resource classification with infill drilling along the periphery and within the currently defined East and West Zones.

Recommendations

Based on the positive results of this preliminary report, SRK recommends that the project be taken to the next level of study. SRK recommends that a preliminary feasibility study ("PFS") be conducted but understands that Hathor wishes to proceed directly to a feasibility study ("FS"). A FS would require the following components:

Field Programs

- Definition drilling of the East and Far East Zones with the goal of taking them to Indicated or Measured classification when a revised estimate is done for the FS;
- Augmentation of the specific gravity database to define the spatial variability of the specific gravity in all resource domains.
- Metallurgical sampling and testing to a FS level;
- Geotechnical drilling of the West, East and Far East deposits, mine accesses and surface facilities;
- Hydrogeological drilling of the West, East and Far East deposits, mine accesses and surface facilities;
- Tailings Management Facility studies;
- Initiation of environmental baseline studies;
- Hydrology studies; and
- Geochemical sampling and testing.

FS Study

- Following the completion of the field program a FS would be completed.
- The cost of the field program and FS would be between $8M to $10M and would take a minimum of 12-18 months.

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1 Introduction

This Preliminary Economic Assessment (“PEA”) Technical Report for the Roughrider Uranium Deposit was compiled by SRK Consulting (Canada) Inc. for Hathor Exploration Limited (“Hathor”). The PEA only considers the East and West Zones which have mineral resource estimates, and does not consider the newly-discovered Far East Zone. Melis Engineering Ltd. (“Melis”) of Saskatoon, Saskatchewan provided all metallurgy and mineral processing-related information for this study.

The Roughrider East and West Zones form part of the Roughrider Uranium Project, an advanced stage uranium exploration project, located in northern Saskatchewan, Canada. The Roughrider Project is located approximately 7 km north of Points North Landing. Hathor owns 100% of the Roughrider Project.

Several sections of this report utilize previous Roughrider technical reports for information and are referenced and signed off by a current Qualified Person (“QP”).

The reader is advised that the preliminary assessment summarized in this technical report is only intended to provide an initial, high-level review of the project potential. The PEA mine plan and economic model include the use of indicated and inferred resources. The inferred resources are considered to be too speculative to be used in an economic analysis except as allowed for in PEA studies. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

The QPs responsible for this report are listed in Table 1.1 along with their responsibilities and site visit dates and descriptions. Each QP in this report takes sole responsibility for their work as outlined in their QP Certificates.

All units in this report are based on the International System of Units (“SI”), except industry standard units. All currency values are Canadian Dollars (“$”).

This report uses abbreviations and acronyms common within the minerals industry. Explanations are located in Section 29.

SRK’s opinion contained herein and effective September 13, 2011, is based on information collected by SRK throughout the course of SRK’s investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This technical report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Hathor, and neither SRK nor any affiliate has acted as advisor to Hathor, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

Table 1.1: Qualified Persons and Site Visit Information

Qualified Person	Responsibility	Site Visit Date	Scope of Site Visit
Gordon Doerksen, P.Eng	Project Management and Economics Report Sections: Executive Summary,1, 2, 14, 17, 18, 20.1.1, 20.2.1, 20.2.4, 21, 24-30	June 27, 2011	- Inspection of selective drill core - General site layout and conditions Diamond drilling operations
Bruce Fielder, P.Eng.	Metallurgy and Mineral Processing Report Section: 12, 16, 20.1.3, 20.1.3	June 27, 2011	- Inspection of selective drill core - General site layout and conditions-Diamond drilling operations
Iouri Iakovlev, P.Eng.	UG Mining Report Sections: 15, 20.1.2, 20.1.4, 20.2.2	No site visit. Site information was transferred from QP Doerksen to QP Iakovlev.	
David Keller, P.Geo.	Geology, and Mineral Resource Estimation Report Sections: 3-11,13, 22	September 13 to 14, 2010 and March 16 to 18, 2011	- Ascertain geological setting; - Inspection of drilling and sampling and exploration practice ; - Witness extent of exploration work; - Examination and logging of core
Mark Liskowich, .Geo.	Permitting, Environmental and Social Considerations Report Sections: 19	No site visit. Extensive knowledge of the area based on numerous visits to the region.	
Bruce Murphy, FSAIMM	Rock Mechanics and Hydrogeology, Geotechnical Considerations Report Sections: 23	June 27, 2011	- Inspection of selective drill core - General site layout and conditions - Diamond drilling operations
Cam Scott, P.Eng	Waste Management, Geotechnical Considerations Report Sections: 17.1	June 27, 2011	- Inspection of selective drill core - General site layout and conditions - Diamond drilling operations

2 Reliance on Other Experts

SRK has not performed an independent verification of land title and tenure as summarized in Section 3 of this technical report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties. SRK relies on Hathor to provide correct information on the land title and tenure of the Roughrider property.

SRK was informed by Hathor that there are no known litigations potentially affecting the Roughrider Project.

SRK utilized the general advice of Greg Newman, P.Eng. of Newmans Geotechnique for the freezing components of this study. SRK takes responsibility for Mr. Newman's advice.

3 Property Description and Location

The Roughrider East and West Zones form part of the Roughrider Uranium Deposit within the RoughrderProject, previously referred to as the Midwest NorthEast Project. The project is located in the eastern Athabasca Basin of northern Saskatchewan, Canada. The project is located approximately 7 km north of Points North Landing, a service centre at the terminus of Provincial Highway 905, approximately 440 km north of La Ronge and approximately 700 km north of Saskatoon. It is located in the Northern Mining District and can be found on 1:50,000 NTS sheet 74-I-08.

The Roughrider Project comprises three contiguous mineral leases (ML-5544, -5545, -5546) and covers an area of approximately 598 ha. The project area has an irregular shape with a north-south dimension of a maximum of 2.5 km along its eastern boundary and east-west dimension of a maximum of 3 km.

The mineral leases adjoin the north-eastern boundary of the Midwest Joint Venture's ("MWJV") (operated by Areva Resources Canada Inc.) mining lease, ML 5264, and are 4.7 km and 1.8 km northeast of Midwest and Midwest A uranium deposits of MWJV respectively. The Dawn Lake uranium deposits are also located approximately 6 km east of the Roughrider Project.

The core camp facility is located within the main lease, on the shore of the northeast bay of McMahon Lake and is located at 556,656 m East and 6,465,610 m North Universal Transverse Mercator NAD83 datum Zone 13 or Latitude 58.3275º North and Longitude 104.0325º West (WGS84 datum).

3.1 Mineral Tenure

The Roughrider Project originally consisted of three contiguous mineral claims, S-107243 staked on January 30, 2004, and S-110759 and S-110760 staked on March 18, 2008, covering a total area of 543 hectares. Hathor carried out a legal survey of the property in 2010. On March 16, 2011, the three mineral claims were converted to mineral leases. Due to minor modification to the eastern property boundary as a result of the legal survey and land tenure changes, the official size of the mineral lease area is 598 ha.

The Roughrider Project currently comprises three mineral leases as listed in Table 3.1 and shown in Figure 3.2. Mineral resources for the Roughrider East and West Zones are contained completely within mineral lease ML-5544. The mineral leases are currently in good standing (Table 3.2).

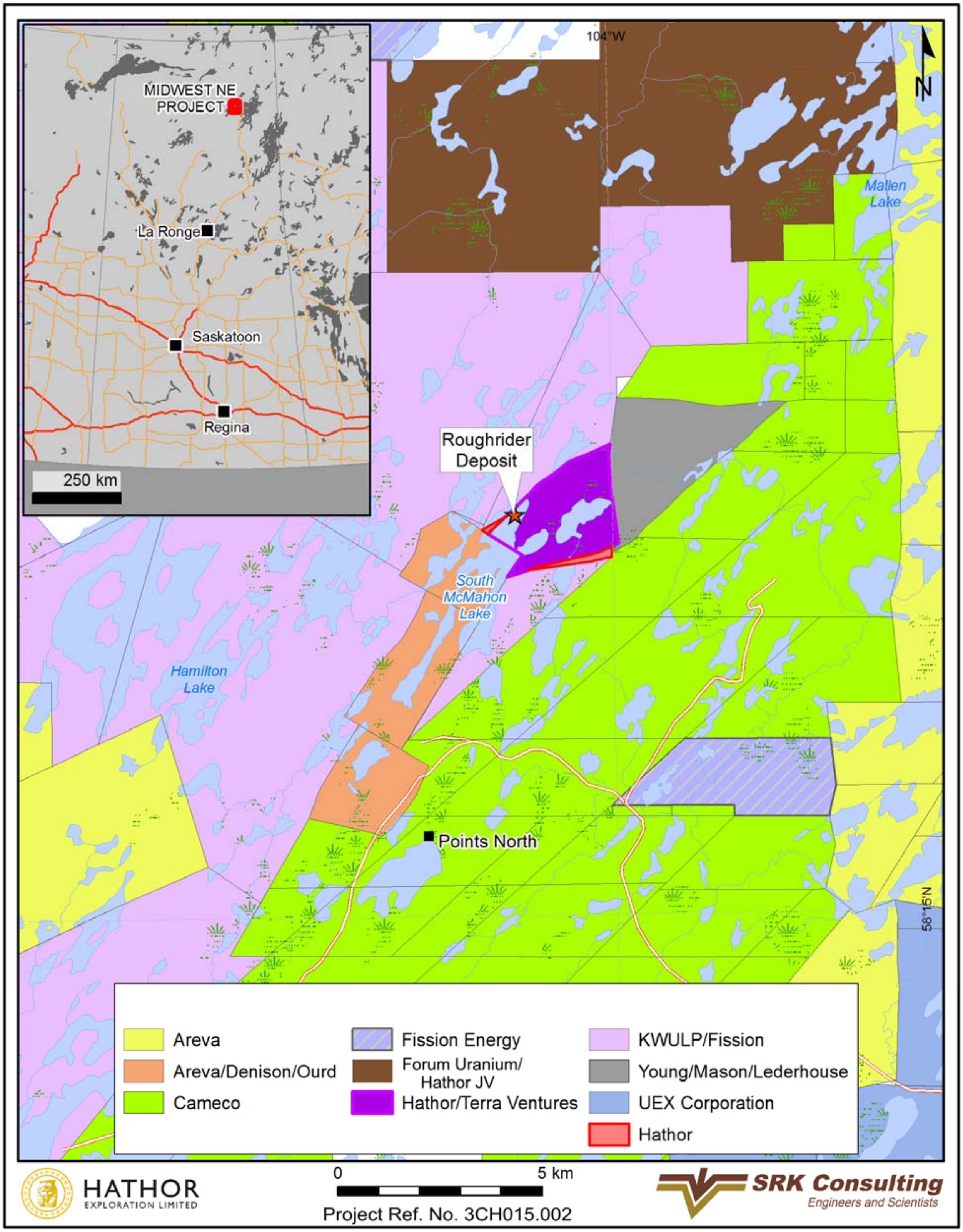

Figure 3.1: Location Map of Roughrider Project, formerly called MidWest NorthEast Project

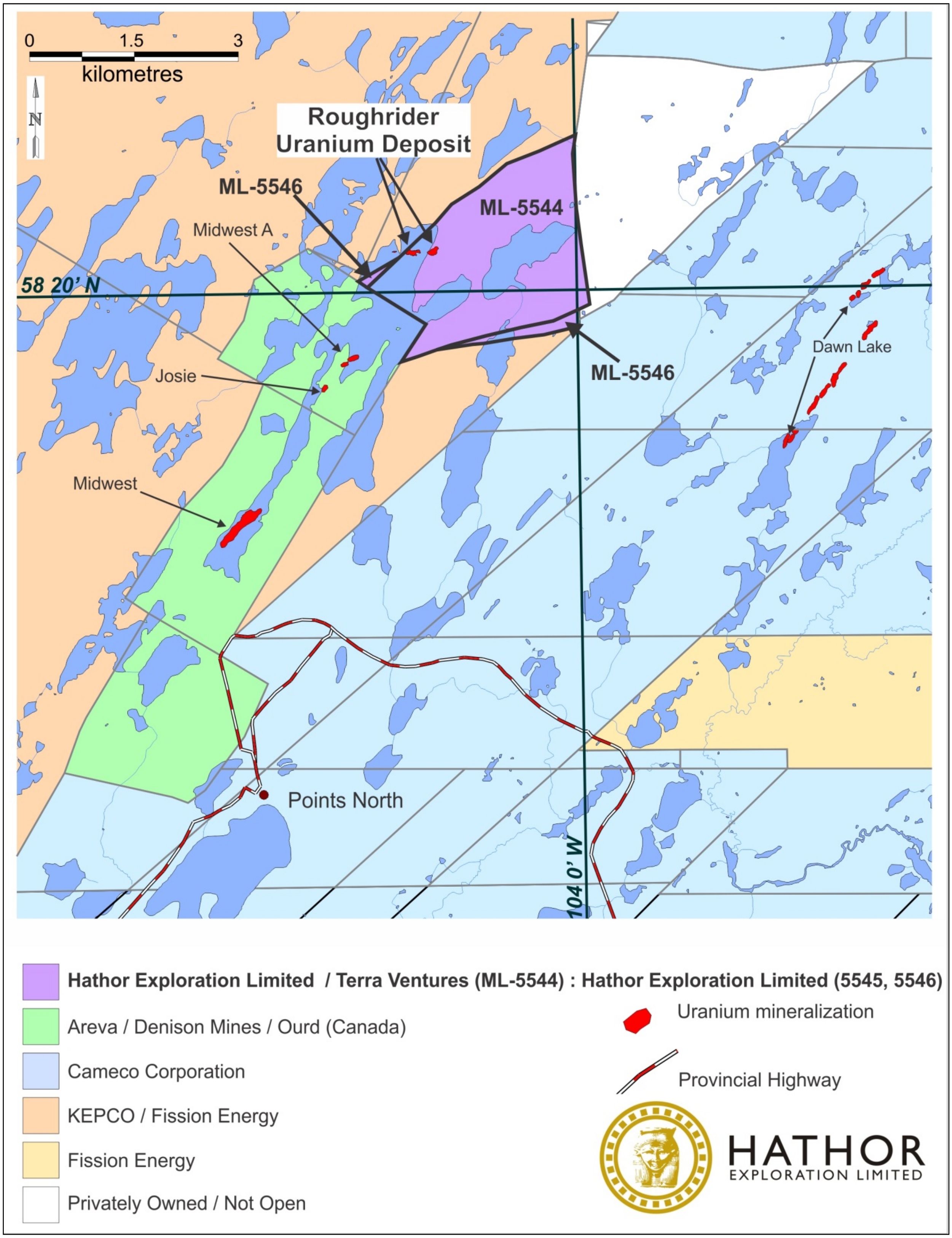

Figure 3.2: Land Tenure Map

Table 3.1: Mineral Tenure Information

Mineral Lease	Previous Mineral Claim	Registered Owner	Date of Record	Size (Ha)
ML-5544	S-107243	Hathor Exploration Limited 90%; Terra Ventures Inc. 10% (effect 100% Hathor ownership – see Section 3.2)	03/16/2011	568
ML-5545	S-110759	Hathor Exploration Limited 100%	03/16/2011	25
ML-5546	S-110760	Hathor Exploration Limited 100%	03/16/2011	5

Table 3.2: Available and Excess Credit Details for Roughrider Project (Saskatchewan Geological Atlas, May 26 2011)

Mineral Lease	Annual Expenditure Requirement ($)	Available Credit ($)	Excess Credit ($)
ML-5544	14,200.00	231,422.00	79,536.58
ML-5545	800.00	5,001.95	-
ML-5546	800.00	6,444.00	216,163.96

3.2 Underlying Agreements

Under an agreement dated September 10, 2004, between Roughrider Uranium Corp. (“Roughrider”) and Bullion Fund Inc. (“Bullion Fund”), Roughrider earned a 90% interest in claim S-107243 (and six other claims that became part of Roughrider’s Russell South property) by paying Bullion Fund an aggregate of $200,000 cash. Bullion Fund retained a ten percent carried interest. On August 10, 2006, Roughrider became a wholly owned subsidiary of Hathor.

A 2% net smelter return on ML-5544 is payable to original Roughrider shareholders.

On April 12, 2007, Terra Ventures Inc. (“Terra”) announced that it had closed a deal with Bullion Fund to acquire an 8% carried working interest in seven claims comprising 56,360 acres in two separate projects located in the Athabasca Basin, Saskatchewan, of which 90% of the remaining 92% working interest was held by Hathor. One of the claims was S-107243. Terra’s interest is carried in all respects through to the completion of a feasibility study and the public announcement that the claims will be put into commercial production. Terra paid $2,300,000 to acquire the interest and also paid a finder’s fee of $69,000.

On March 24, 2008, Terra announced that it had closed its agreement with Bullion Fund to purchase Bullion Fund's remaining 2% carried working interest in the Roughrider Project of Hathor. This purchase increased Terra's holding to a 10% carried working interest through to the completion of a feasibility study and the public announcement that the claims will be put into commercial production. The consideration paid by Terra to acquire this interest was $2,500,000 and 3,000,000 shares of Terra.

On April 18, 2011, Hathor and Terra announced that they had executed a binding letter agreement pursuant to which Hathor will acquire, in an all-share transaction, all of the issued and outstanding shares of Terra. On May 9, 2011, Hathor and Terra announced that they had executed a definitive plan of arrangement agreement ("the Arrangement") to complete the previously announced merger. This will result in consolidation of 100% ownership of the Roughrider Project.

On August 2, 2011, Terra received approval from 96% of votes cast at a special meeting of its shareholders in Vancouver. On August 4, 2011, Terra received final approval from the Supreme Court of British Columbia to complete the Arrangement. On August 5, 2011, Hathor and Terra announced the completion of the plan of arrangement. The Arrangement became effective on August 5, 2011 and Terra is now a wholly-owned subsidiary of Hathor.

3.3 Permits and Authorization

All the necessary permits for surface exploration on the property are in place and current.

3.4 Environmental Considerations

Activities on the project property to date have been limited to resource delineation and gathering of environmental baseline data. The environmental liabilities associated with these activities are consistent with low impact exploration activities. The mitigation measures associated with these impacts are accounted for within the current surface exploration permits and authorizations.

3.5 Mining Rights in Saskatchewan

Exploration and mining in Saskatchewan is governed by the Mineral Disposition Regulations 1986, and administered by the Mines Branch of the Saskatchewan Ministry of Energy and Resources.

There are two key land tenure milestones that must be met in order for commercial production to occur in Saskatchewan: (1) conversion of a mineral claim to mineral lease, and (2) granting of a Surface Lease to cover the specific surface area within a mineral lease where mining is to occur.

A mineral claim does not grant the holder the right to mine minerals except for exploration purposes. Subject to completing necessary expenditure requirements, mineral claims can be renewed for a maximum of twenty-one years.

Beginning in the second year, and continuing to the tenth anniversary of staking a claim, the annual expenditure required to maintain claim ownership is twelve dollars per hectare.

A mineral claim in good standing can be converted to a mineral lease by applying to the mining recorder and have a completed boundary survey.

In contrast to a mineral claim, the acquisition of a mineral lease grants the holder the exclusive right to explore for, mine, recover, and dispose of any minerals within the mineral lease. Mineral leases are valid for ten years and are renewable.

Land within the mineral lease, surface facilities and mine workings are considered to be located on Provincial lands and therefore owned by the Province. Hence, the right to use and occupy those lands is acquired under a surface lease from the Province of Saskatchewan. A surface lease is issued for a maximum of 33 years, and may be extended as necessary to allow the lessee to operate a mine and/or plant and undertake reclamation of disturbed ground.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

The property can be accessed either by helicopter, fixed wing aircraft or winter road from Points North Landing. Points North Landing is accessed from La Ronge via provincial Highway 102 to South End and then Provincial Highway 905. The nearest, sizable population centre is La Ronge, approximately 440 km by road to the south. There is a daily commercial airline service from Saskatoon to Points North Landing

4.2 Local Resources and Infrastructure

La Ronge is the nearest commercial/urban centre where most exploration supplies and services can be obtained. Mining personnel could be drawn from the general Saskatchewan populace that supplies the existing uranium mining operations. Two airlines offer daily, scheduled flight services between Saskatoon, La Ronge, Prince Albert, and Points North Landing.

Electrical power is available from the provincial grid through a switching station at Points North Landing. Adequate water for a mining operation could be obtained from the lakes on the property.

4.3 Climate

The climate is a mid-latitude continental climate, with temperatures ranging from 33º C in the summer to -45ºC in the winter. Winters are long and cold, with mean monthly temperatures below freezing for seven months of the year. Annual precipitation is about 500 mm per year, with half of that in the summer months. Winter snow pack averages 70 cm to 90 cm. Lake ice forms by mid-October and usually melts by mid-April. Field operations are possible year round with the exception of limitations imposed by lakes and swamps and the periods of break-up and freeze-up.

4.4 Physiography

The property is characterized by a relatively flat till plain with elevations ranging from 477 metres (South McMahon Lake) to 502 metres above mean sea level (Figure 4.1). Throughout the area, landforms distinctively trend northeast arising from passages of glacial ice from the northeast to the southwest or the property. The Roughrider East Zone is located on the flank of a glacial drumlin while the West Zone lies under South McMahon Lake. About 60% of the property is land and 40% water. Vegetation consists mainly in the form of small jack pine and black spruce reaching heights of approximately 8 m.

Figure 4.1: Typical Landscape in the Project Area

5 History

5.1 Prior Ownership

The mineral claim S-107243 was acquired by Bullion Fund on January 30, 2004. Bullion Fund sold a 90% interest in the property to Roughrider on September 10, 2004 and retained a 10% carried interest. On August 10, 2006, Roughrider became a wholly owned subsidiary of Hathor. On April 12, 2007, Terra purchased an 8% carried working interest from Bullion Fund for seven claims comprising 56,360 acres over two separate projects (Midwest Northeast and South Russell) in the Athabasca Basin. The claims included mineral claim S-107243. On March 24, 2008, Terra announced that it had purchased Bullion Fund's remaining 2% carried working interest in mineral claim S-107243.

5.2 Previous Exploration Work

Between 1969 and 1974, following the discovery of the Rabbit Lake uranium deposit in 1968 by Gulf Minerals Ltd., Numac Oil and Gas ("Numac") held the large Permit Number Eight over the Midwest Lake (McMahon Lake) and Dawn Lake areas. Prospecting, airborne radiometric surveys and lake sediment sampling for uranium and radon were carried out in 1969 and 1972 (Forgeron, 1969; Beckett, 1972). At the time, Numac, in conjunction with their partners Esso Minerals and Bow Valley Industries, focused on the Midwest Lake area, located adjacent to Hathor's Roughrider Project.

In 1976, Asamera Oil Corporation ("Asamera") initiated the Dawn Lake project, located approximately six kilometres southeast of the current Roughrider Project. Asamera discovered the Dawn Lake 11, 11A, 11B, and 14 zones in 1978.

In 1983, the Saskatchewan Mining and Development Corporation ("SMDC"), predecessor to Cameco Corporation ("Cameco") became the operator of the Dawn Lake Joint Venture. By 1995, the Dawn Lake Joint Venture consisted of Cameco, Cogema Resources Inc. (now Areva Resources Canada Inc.), PNC Exploration Canada Ltd., and Kepco Canada Ltd. (Jiricka et al., 1995). The Dawn Lake Joint Venture held the Esso North claim until it lapsed in 2003.

Early work by Asamera on the Esso North claim consisted of INPUT-electromagnetic ("EM") and aeromagnetic surveys in 1977, followed by airborne very low frequency (VLF)-EM, magnetic and radiometric surveys in 1978 and 1979 by Kenting and Geoterrex, respectively. These surveys located an east-west trending conductor of moderate strength and a radiometric anomaly associated with a broad VLF-EM response on the eastern portion of the Esso North claim (Parker, 1982).

From 1978 to 1981, Turam, Vector Pulse EM, and VLF-EM surveys confirmed the east-west conductor as well as some weaker northeast trending VLF-EM conductors. The east-west conductor occurs just outside the western boundary of Hathor's lease ML-5544.

During this same period, Asamera drilled 21 holes on the Esso North claim (Parker, 1982; Asamera, 1982). The first ten holes, EN-1 to EN-10, were drilled across the projected northeast strike extent of the Roughrider Project. These holes are located within Hathor's lease ML-5544 and penetrated basement rock for an average length of 25 m.

The other eleven holes were drilled on the main east-west striking conductor. Results, however, were discouraging; the highest radioactivity was encountered in drillhole EN-14 with 590 counts per second ("cps") on a radiation detector. Basement lithologies intersected in drillholes included Archean granitoid, pegmatite, migmatite, and rare pelitic gneiss. Some evidence of structural disturbance and alteration was observed in the Athabasca sandstone intersected in drillholes EN-14, EN-15, and EN-16. Parker (1982) recommended relogging of the drill core to determine if any structural features had been missed. Only EN-14 and EN-15 are collared within the current Hathor lease ML-5544.

In 1984, SMDC carried out Time Domain EM ("TEM") on the Esso North claim and completed two additional holes (Roy et al., 1984). Drillhole EN-18 targeted a weak TEM conductor in the vicinity of the east-west conductor. Results of this hole were negative. Drillhole EN-19 targeted a weak northeast trending TEM conductor. It intersected faulting and alteration in the Athabasca sandstone, but no other interesting features, and ended in pegmatite. Drillholes EN-18 and EN-19 are located within Hathor's lease ML-5544.

Exploration on the Esso North claim was dormant until 1995 (Jiricka et al., 1995), when Cameco resurveyed the area with TEM and located both the east-west conductor and the weak northeast striking conductor. The latter target was tested by one hole, EN-20; it intersected faulted and altered sandstone but no significant radioactivity. The basement consisted of granite, pegmatite, as well as minor pelitic and psammitic gneiss.

Radioactivity of up to 379 cps occurred in the basement, but the cause of the conductor was not found. Illite content increased with depth. Above-background uranium and lead values occurred in the sandstone. No significant trace element enrichment was noted in the basement. Hole EN-20 is located within lease ML-5544.

In 1996 one drillhole, EN-21, was completed that targeted the east-west conductor. This conductor is located just west of Hathor's property. No conductive material was intersected and the basement lithology was granite. Anomalous lead values present were attributed to heavy minerals in the sandstone. The lower forty percent of the sandstone column was bleached (Jiricka et al., 1996).

5.3 Historical Mineral Resource and Mineral Reserve Estimates

There are no historical mineral resource and/or mineral reserve estimates for the Roughrider West or East Zones prior to the acquisition of the property by Roughrider Uranium Corporation.

5.4 Historical Production

There has been no historical production from the Roughrider West or East Zones.

6 Geological Setting and Mineralization

The most significant uranium district in Canada is the Athabasca Basin, which covers over 85,000 km^2 in northern Saskatchewan and north eastern Alberta. The saucer-shaped basin contains a relatively undeformed and unmetamorphosed sequence of Mesoproterozoic clastic rocks known as the Athabasca Group. These rocks lie unconformably on the deformed and metamorphosed rocks of the Western Churchill Province of the Archean Canadian Shield. The basement rocks consist of Archean orthogneisses, which are overlain by, and structurally intercalated with, the highly deformed supracrustal Palaeoproterozoic Wollaston Group (Annesley et al., 2005).

The Athabasca Basin is elongated along an east-west axis and straddles the boundary between two subdivisions of the Western Churchill Province. The Rae Subprovince to the west and the Hearne Subprovince to the east. The subprovinces are separated by the northeast trending Snowbird Tectonic Zone, locally known as the Virgin River-Black Lake shear zone in the area of the Athabasca Basin.

The Hearne Craton beneath the eastern Athabasca Basin comprises variably reworked Archean basement, which is dominated by granitic domes and foliated to gneissic granitoid rocks with infolded outliers of Paleoproterozoic metasedimentary rocks. The structural and tectonic regime of the area has been influenced strongly by collisional tectonics between the Hearne and Superior Cratons during the early Proterozoic Trans-Hudson Orogen, which occurred approximately 1.9 to 1.77 billion years ago(“Ga”).

Prior to deposition of the Athabasca Group, rocks of the Rae and Hearne Provinces that would later form the basement of the basin rocks experienced a lengthy period of weathering and non-deposition. As a consequence, the basal Athabasca stratigraphy is underlain by a regolith of deeply weathered, hematite-stained basement. In places, the preserved regolith can reach a thickness of 50 m. Typical thicknesses range from approximately five to ten metres. On a regional scale, the unconformity surface is relatively flat except where disrupted by post-Athabasca faulting, or where local paleotopography has been identified.

Unconformably overlying the basement rocks is the late Mesoproterozoic Athabasca Group consisting mainly of fluvial clastic sedimentary rocks, which are about 1,400 m thick in the central part of the basin (Ramaekers, 1990).

The Athabasca Group comprises eight formations in which four broadly fining-upward, unconformity-bound cycles can be distinguished. Overall, sedimentary provenance was from the east, south, and northwest. Lithologies are dominated by fine- to coarse-grained, partly pebbly or clay-intraclast-bearing quartz arenites. Minor conglomerates, mudstones, and dolostones also occur.

Apart from faulting and local folding associated with thrusting, the Athabasca Group strata are undeformed and unmetamorphosed. Age dating of zircons and diagenetic fluorapatite (SGS, 2003) indicate an age of sedimentary deposition around 1.77 Ga, post-dating the Trans-Hudson Orogeny (circa 1.9 Ga to 1.77 Ga).

In the eastern Athabasca Basin, the Manitou Falls Formation is the only formation present. It is subdivided into four units, from bottom to top, designated MFa to MFd. These are dominantly fluvial sandstones with some interbedded conglomerates.

Recent scientific publications have suggested a reclassification of the MFa to its own separate unit called the Read Formation. In the area many exploration geologists still use the term MFa (Jefferson et al., 2007).

6.1 Property Geology

6.1.1 Hearne Subprovince

Four important lithostructural domains have been identified in the Hearne Subprovince: the Eastern Wollaston Domain (“EWD”), Western Wollaston Domain (“WWD”), Wollaston-Mudjatik Transition Zone (“WMTZ”), and Mudjatik Domain (“MD”) (Annesley et al., 1997; Annesley et al., 2005). The basement rocks within the property are part of the WMTZ. The majority of the uranium occurrences and all currently producing uranium mines in the area are hosted in rocks of the WWD and WMTZ. Certain lithologies, coupled with the deformational history of some domains, have had a strong influence on the location of the Athabasca unconformity-type uranium deposits.

The MD is distinguished from the adjacent northeast trending “straight” belts of the WWD by having an arcuate map pattern resulting from regional dome-and-basin fold interference attributed to the Trans-Hudson Orogen. The MD is interpreted to have been less strongly affected by Hudsonian transpressive tectonics and is more deeply eroded. Hence, the occurrence of Paleoproterozoic supracrustal rocks is relatively limited in this domain. The eastern boundary with the WWD is arbitrarily drawn along the transition into mainly northeast-trending structures. The MD is dominated by variably migmatitic Archean orthogneisses; however, it also incorporates minor infolded supracrustal outliers of the Wollaston Supergroup.

Model ages from the orthogneiss indicate a crustal history beginning as early as 3.6 Ga with extensive crust development approximately 2.92 Ga. Pelitic to psammitic supracrustal rocks and mafic granulites, minor quartzites, calc-silicates, marbles and ultramafic rocks, as well as rare oxide, silicate and sulphide facies iron formations occur in narrow arcuate bands throughout, defining the dome-and-basin pattern.

In the east, most of these supracrustal remnants have been correlated with the Wollaston Supergroup. Metamorphic grades range from upper amphibolite to granulite facies (Annesley et al., 2002; SGS, 2003).

Away from the Roughrider Uranium Deposit within the Roughrider Project area, the reddish to greenish paleoweathering profile immediately below the sub-Athabasca unconformity is variable in its development but typically extends to a depth of ten metres to thirty-five metres. It comprises a thin (less than 1 m) zone of bleached rock that is typically illitic to kaolinitic in composition. Immediately beneath is a zone of variably developed hematite alteration (red zone). This is separated from the lowermost alteration zone, the chlorite-altered green zone, by a transitional red-green zone, which is a combination of hematite and chlorite alteration. Within the Roughrider Uranium Deposit, the paleoweathered regolith is overprinted and obliterated by hydrothermal alteration. In some cases, however, a ghost clay signature of the kaolinitic zone is still evident.

6.1.2 Athabasca Group

The property is underlain by 195 to 215 m of sandstone belonging to the Manitou Falls Collins Member (MFc) and Bird Member (MFb) of the Athabasca Group. The Read Formation (MFa) is missing. The MFc can reach a thickness of seventy to 100 m and is composed of a fine-grained, homogeneous, beige to maroon sandstone. The MFb member ranges from 100 m to 130 m in thickness and comprises a heterogeneous mix of sandstone, pebbly sandstones and conglomerates. The conglomerates include a distinctive “Marker Conglomerate” that can be correlated regionally. The basal conglomerate is not ubiquitous throughout the property; in places immediately overlying the Roughrider Uranium Deposit it may be absent. Typically the unconformity is approximately 196 m to 221 m below the surface.

6.1.3 Surficial Geology

The Athabasca Basin and surrounding areas bear the strong imprint of Quaternary glaciation. During the Pleistocene Epoch, the northern half of Saskatchewan was scoured by the Laurentide ice sheet that was generally moving in a south-westerly direction. Over the Athabasca Basin, glacial erosion of the less resistant sandstone resulted in an increased sediment load in the ice. Consequently, the glacial drift cover is much more extensive and thicker over the basin than the rest of the shield region (SGS, 2003).

The surficial geology within the property is characterized by portions of two low drumlins trending in a north-east direction. The drumlin tops are approximately 200 m to 50 m above local lake surface. The glacial deposits are composed generally of a sandy till that contains primarily reworked Athabasca sand grains, cobbles and boulders.

No outcrops have been observed. Drilling has encountered overburden depths between nine and twelve metres. In the vicinity of the Roughrider Uranium Deposit, McMahon Lake has a water depth of between five and twelve metres.

6.2 Deposit Geology

The Roughrider Uranium Deposit overlies the Wollaston-Mudjatik Transition Zone (“WMTZ”) of the Wollaston Domain. The basement is structurally complex, comprising steeply dipping Wollaston Group rocks interfingering Archean granitic to granodioritic orthogneisses. Interpretations of aeromagnetic data suggest that several Archean granitic domes dominate the basement geology.

Several basement packages are recognized in the vicinity of the Roughrider Uranium Deposit and include the:

- Wollaston Group;
- Hanging Wall Wedge (“HWW”);
- Foot Wall Wedge (“FWW”); and
- Midwest Dome (“MWD”).

The property geology indicates that the Roughrider Project is situated in the basal part of the Wollaston Group, which is dominated by garnet- and cordierite-bearing pelitic gneisses with subordinate amounts of graphitic pelitic gneisses and psammopelitic to psammitic gneisses, and rare garnetites.

The pelitic gneiss varies from equigranular to porphyroblastic in texture. The porphyroblasts vary in size up to centimetre-scale and normally comprise red almandine-rich garnets when fresh. The variation in size and concentration of garnets allows the identification of several units that are termed the "Garnet Marker Gneiss”. The rocks are variably foliated and often micro-scale folding can be observed.

The very high abundance of garnet within the gneissic rock package is interpreted to be a direct result of the highly aluminous nature of the precursor sediments. The gneisses have been intruded by syn- to post-peak metamorphic felsic pegmatites, granites, and microgranites of Hudsonian age. These rocks locally contain up to 400 parts per million (“ppm”) of primary uranium.

Proximal to mineralization, graphite in graphitic pelitic gneisses has been consumed by alteration and mineralization; distal to mineralization, the graphite appears to be discontinuous. These two features may help explain the absence of basement-hosted graphitic conductors at the Roughrider Uranium Deposit or elsewhere on the property.

Both the FWW and HWW are complex packages and comprise variable amounts of granitic to tonalitic orthogneiss that was subjected to local anatexis. The gneiss was intruded by younger pegmatites, leucogranites and microgranites. From drilling data, the FWW is only locally present and has been interpreted to plunge to the west and does not extend from the Roughrider West Zone eastward across to the Roughrider East Zone. It is possible that several FWW are present, and these represent slivers of either the MWD or the FWW that been carved off during deformation.

The MWD comprises strongly foliated orthogneisses that range in composition from granitic to dioritic. They additionally contain volumetrically minor amounts of partially melted material, and younger 'Hudsonian' pegmatites, leucogranites and microgranites.

Hydrothermal calc-silicate alteration of the orthogneisses is present locally. The alteration is interpreted to be post-peak metamorphism in age and is probably related to the introduction of the Hudsonian felsic rocks. The sandstone and basement rocks have been subjected to several episodes of brittle deformation, including the brittle reactivation of older ductile shear zones

6.3 Structural Geology

Macro-scale geophysical, geological and structural modelling suggests that Roughrider Project is cross cut by a large number of structures. The two main structures to note are: (1) the east-west trending Roughrider Corridor, which hosts the Roughrider West Zone, and (2) the northern extension of the north-northeast trending Midwest Trend, that hosts the Midwest and Midwest A Deposits on the adjacent mineral leases to the south of the MidWest Northeastproject.

The Roughrider West Zone is centered on the Roughrider Structure, while the Roughrider East Zone is centred at or near the intersection of the Midwest Trend and Roughrider Corridor shown in Figure 6.1.

6.4 Mineralization

To date, core drilling on the Roughrider Project has focussed on mineralization at the Roughrider Uranium Deposit, which comprises three zones, the West, the East, and the Far East Zones (Figure 6.2).

The Roughrider West Zone was discovered during the winter drilling program of February 2008. A hydrothermal clay alteration system was intersected in drillhole MWNE-08-10, while high-grade uranium mineralization (5.29% uranium oxide (“U_3O_8”) over a core length interval of 11.9 m) was intersected in drillhole MWNE-08-12.

The Roughrider East Zone was discovered during the summer drilling program in September 2009. Hydrothermal alteration was intersected in a number of earlier drillholes during the summer program. High-grade uranium mineralization (12.71% U_3O_8over a core length interval of 28.0 m) was intersected subsequently in drillhole MWNE 10-170. This zone was delineated by drilling during the winter and summer of 2010. The best intersection to date was obtained in drillhole MWNE-10-648, which intersected 7.75% U_3O_8 over a core length interval of 63.5 m.

A third zone, the Roughrider Far East Zone, was discovered during the winter drilling program in February 2011. The discovery drillhole intersected 1.57 % U_3O_8 over core length of 37.5 m. The current outline of the Far East Zone is defined by mineralization in 28 of 40 drillholes completed in the immediate vicinity of Roughrider Far East Zone; weak mineralization in other drillholes is not included in the current outline of the Far East Zone. The best intersection to date is drillhole MWNE-11-715, which intersected 7.91% U_3O_8 over a core length interval of 27.0 m. No mineral resources are currently available for this zone and included in this study. The Far East Zone covered greater detail in section 9.1.3.

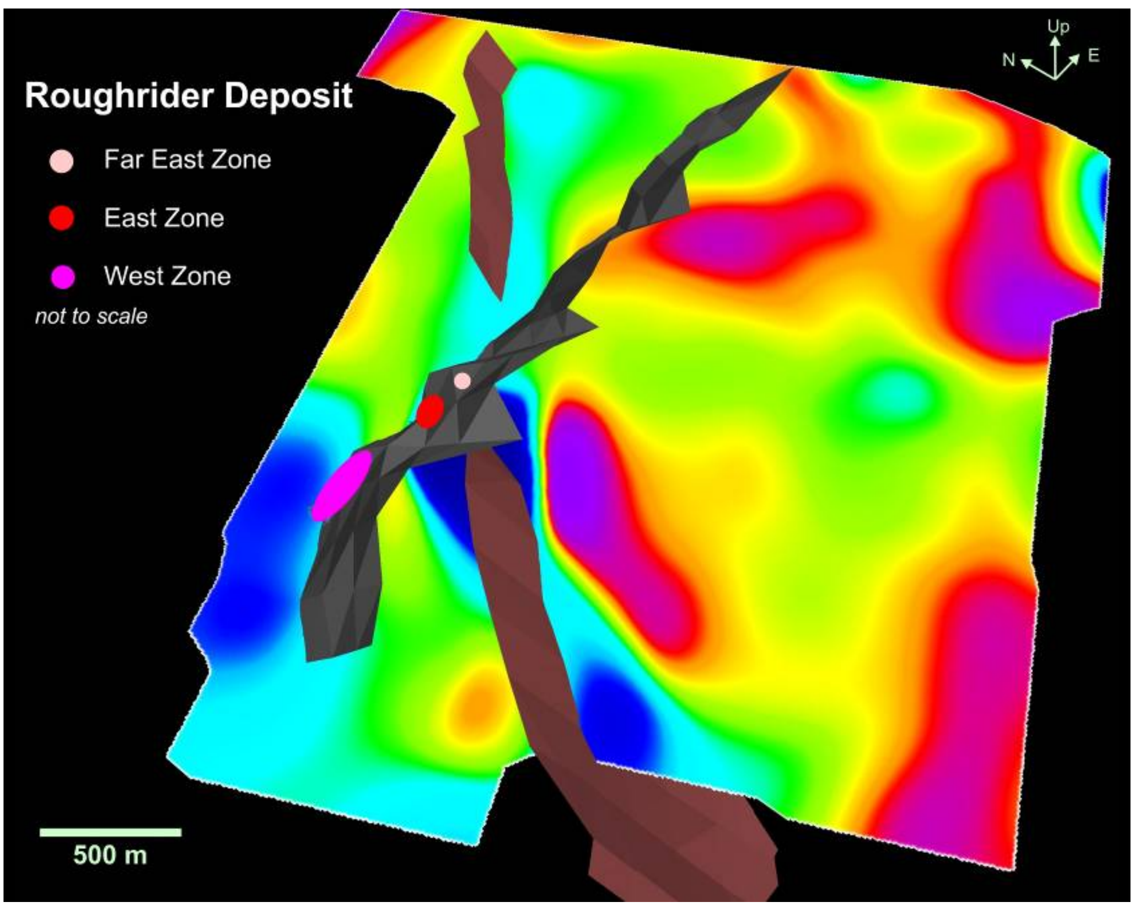

Figure 6.1: Structural Interpretation in the Roughrider Project Superimposed on Resistivity Depth Slice Plan (approximately 250 m depth)

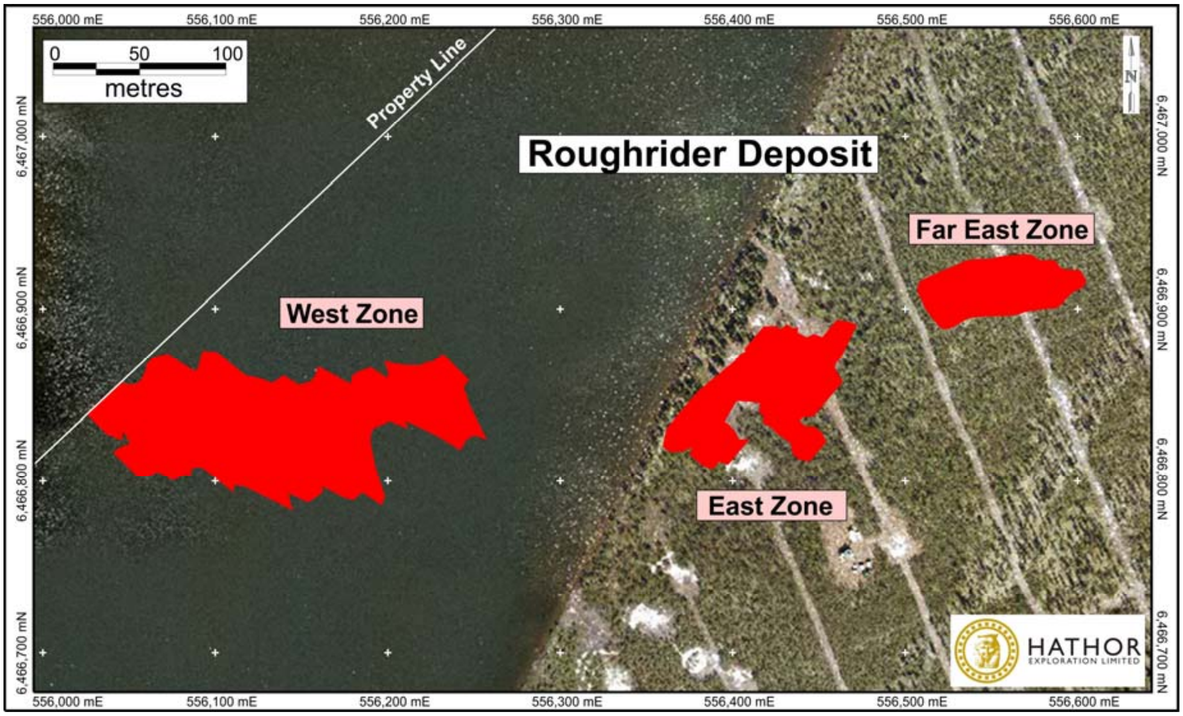

Figure 6.2: Surface Projection of Roughrider West, East and Far East Zones

Keller et al. (2011a) and McCready et al. (2009) describe the alteration and mineralization of the Roughrider West Zone in greater detail. Keller et al. (2011b) describe the alteration and mineralization of the East Zone.

The Roughrider West Zone is defined by approximately 149 diamond drillholes (123 of which were used to evaluate the mineral resource), and has been intersected along a northeast-southwest strike length of approximately 200 m with an across-strike extent of 100 m. Uranium mineralization occurs at depths of 190 m to 290 m below surface. Mineralization in the West Zone is confined to an east west trending corridor of deformation that dips to the north. The uranium mineralization at the Roughrider Deposit is hosted predominantly within basement rocks. Only minor amounts of uranium occur at or above the unconformity.

The Roughrider East Zone is currently defined by approximately 88 diamond drillholes, (21 of which were used to evaluate the mineral resource), and has a surface projection of approximately 120 m long in a north-easterly direction, which corresponds to a down-dip length of approximately 125 m, and an across-strike extent of up to 70 m. Uranium mineralization has a vertical extent of up to eighty to 100 m, starting at depth approximately 250 m from surface, and some 30 m to 50 m below the unconformity. This is slightly deeper than the Roughrider West Zone.

The Roughrider East Zone is located within the same East-West deformation corridor as the Roughrider West and Far East Zones known as the Roughrider Corridor. Based on recent resistivity and structural modelling, the Roughrider East Zone is centred at or near the intersection of the Midwest Trend and the Roughrider Corridor.

Mineralization forms moderately dipping, cigar-shaped shoots along the intersection of these two controlling structures. Uranium mineralization at the Roughrider East Zone is exclusively hosted within basement rocks, and has been intersected within altered rocks of both the Wollaston Group and in syn- to post-tectonic Hudsonian igneous rocks, including the overlying HWW and is developed atop the Midwest Dome of Archean granitic gneiss.

As currently modelled, the West Zone occurs as a number of high grade lenses (greater than 3.0% U_3O_8mantled by an envelope lower grade mineralization that is defined by a 0.05 % U_3O_8cut off. The mineralization plunges moderately to the north or northwest. The contacts between these mineral lenses are sharp. In contrast, the East Zone comprises a series of stacked, parallel lenses of high grade material defined by 0.4% U_3O_8that dip and plunge to the northeast. Unlike the West Zone, the lenses are not uniformly mantled by an extensive rim of low grade mineralization. Abundant low grade mineralization is present outside of the modeled mineral lenses in the current resource model, and represents additional resource potential.

A graphical depiction of the relationship between bedrock geology, mineralization, structure and alteration at the Roughrider West and East Zones is represented in Figure 6.3 and Figure 6.4. Each section shows the uranium mineralization hosted in a corridor or wedge of Wollaston Group rocks, literally sandwiched between and extending into the MWD to the southeast and the HWW on the northwest.

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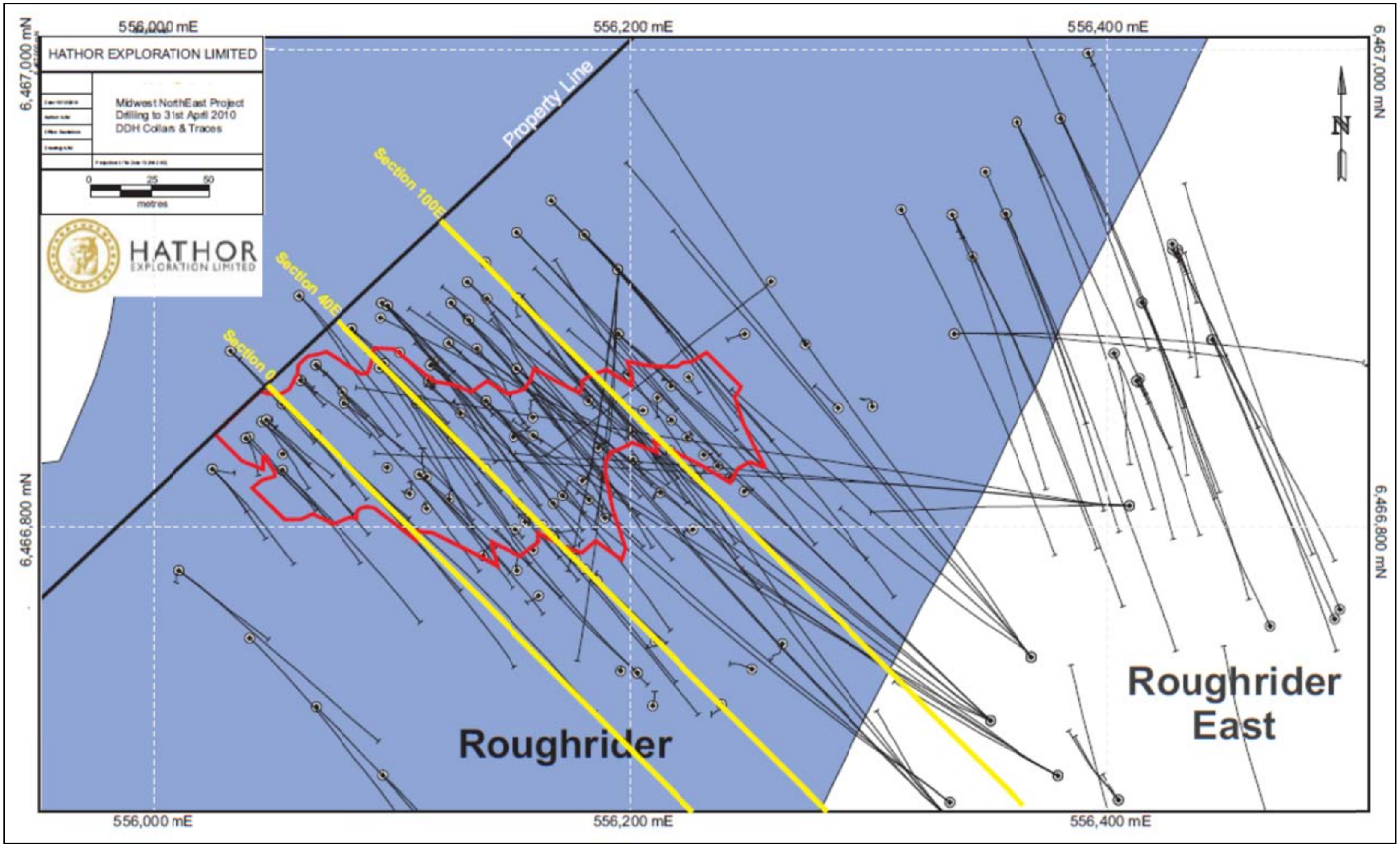


Figure 6.3: Location of Section Lines Roughrider West Zone

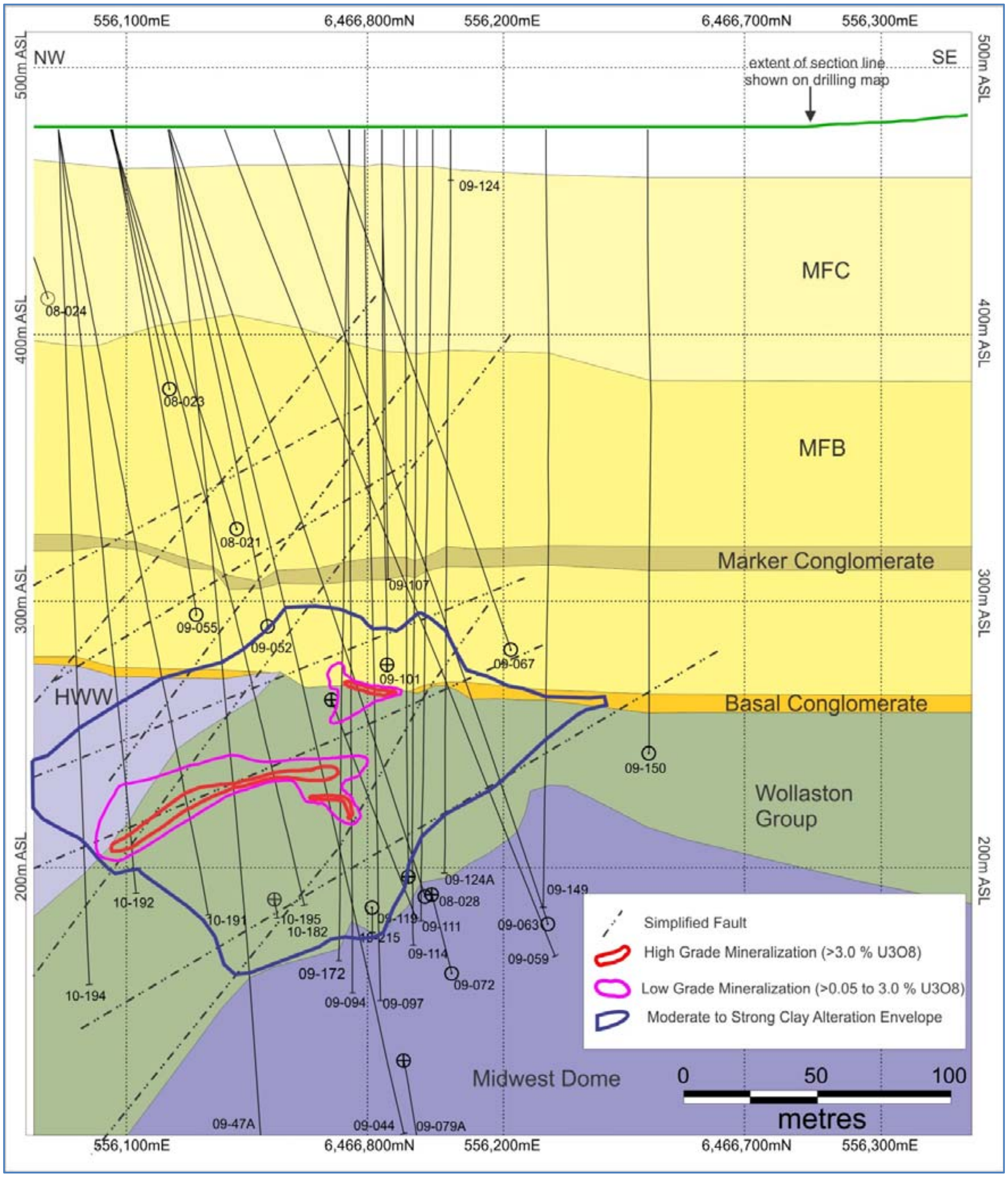


Figure 6.4: Geological Cross Section Roughrider West Zone, Section 40 East (Hathor, 2011)

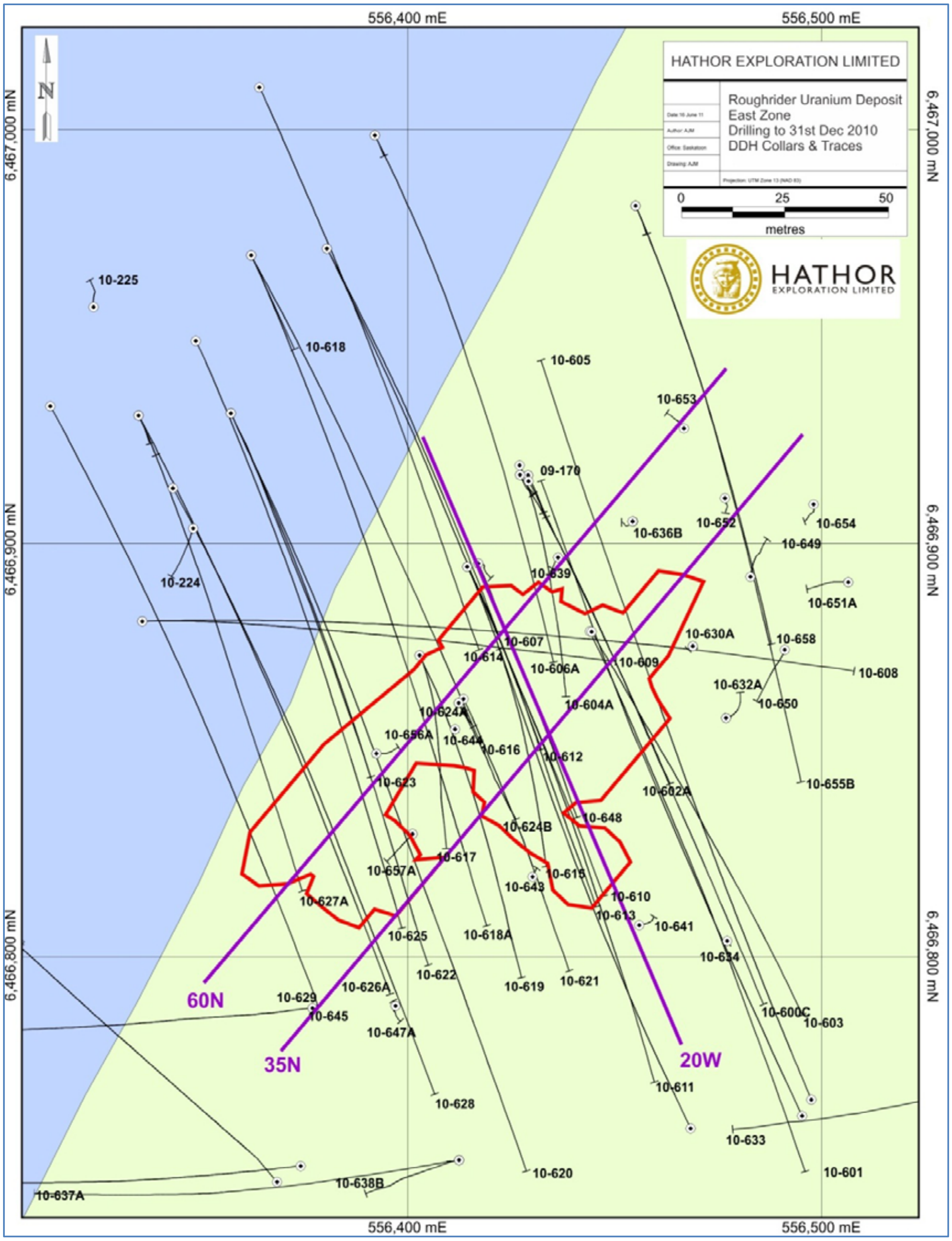


Figure 6.5: Location of East Zone Section Lines

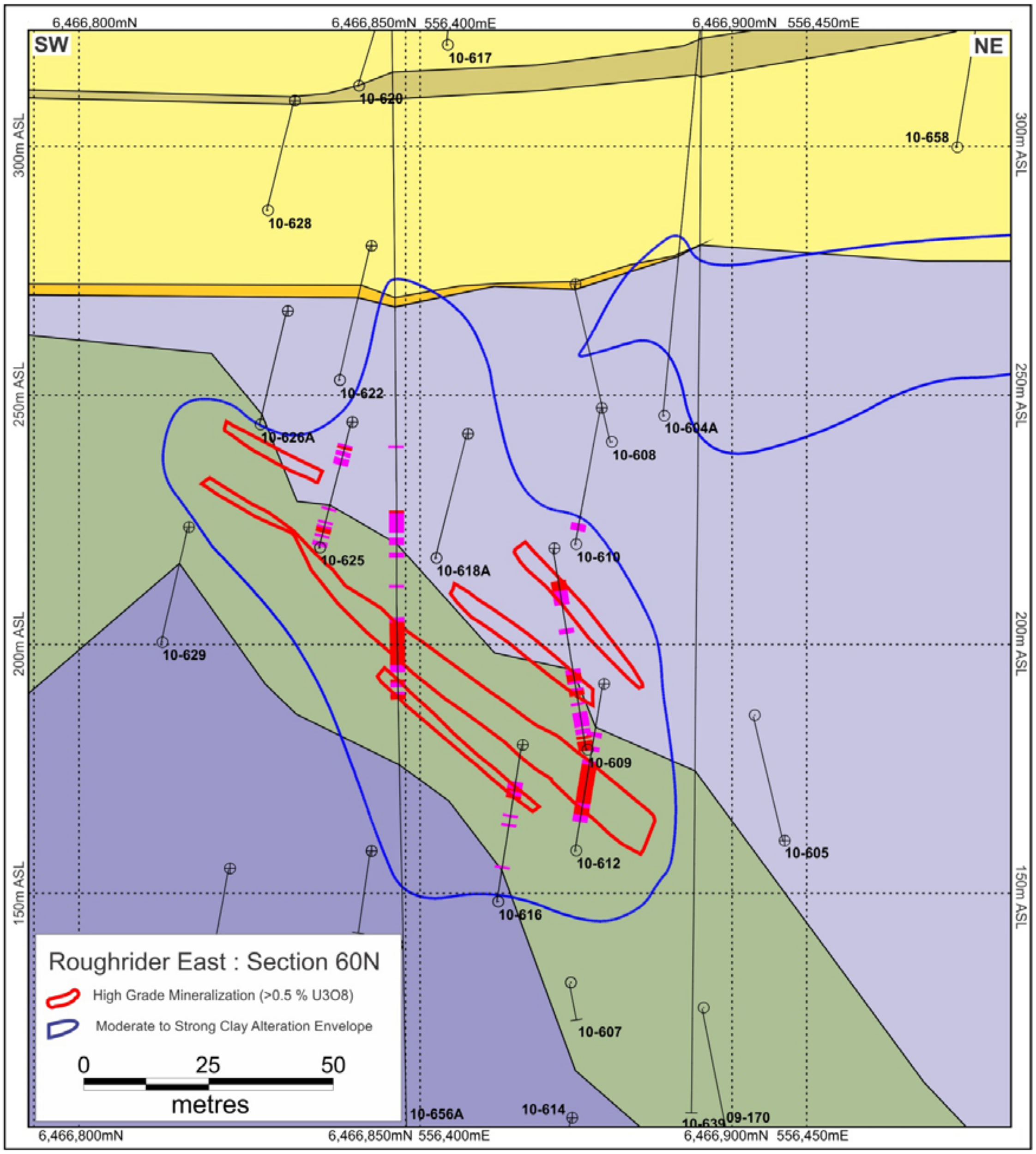


Figure 6.6: Geological Cross Section Roughrider East Section 60 North. (Hathor, 2011)

Uranium mineralization is highly variable in thickness and style in all zones. High grade mineralization occurs primarily as medium- to coarse-grained, semi-massive to massive pitchblende with what has been termed worm-rock texture, and texturally complex redox-controlled mineralization. This high grade mineralization is intimately associated locally with lesser amounts of red to orange coloured oxy-hydroxillized iron oxides. Yellow secondary uranium minerals, probably uranophane, are present locally as veinlets or void-filling masses within the high grade primary mineralization.

Lower grade mineralization occurs as either disseminated grains of pitchblende, fracture-lining, or veins of pitchblende.

Galena occurs in a number of habits and is variably present in the uranium mineralization. The lead is presumed to have formed from the radioactive decay of uranium. Veinlets of galena are up to 5 ml thick and either crosscut massive pitchblende, as anhedral masses (less than 1 ml in size) interstitial to the massive pitchblende, or as fine-grained, sub-millimetre-scale disseminated flecks of galena omnipresent throughout mineralized drill core. In all cases, the galena appears to have formed later than the uranium mineralization.

Mineralization is in general terms, mono-metallic in composition. In the West Zone, visible, crystalline nickel-cobalt sulpharsenides are present locally in a number of drillholes (for example in holes, MWNE-08-12, MWNE-08-19, MWNE-08-37, MWNE-096-56, MWNE-09-90, MWNE-09-101, MWNE-10-188B, MWNE-10-210, MWNE-10-216). At the East Zone, the presence of nickel-cobalt sulpharsenides is rare. Only three individual sample assays returned values of greater than one percent arsenic, nickel or cobalt. Most samples contain less than 0.1% of these elements. The exact relationship of these elements to uranium is variable and still unclear at this time. However, unlike many unconformity-type uranium deposits in the Athabasca Basin, variable amounts of copper mineralization are present within the Roughrider Deposit. Individual samples locally return values greater than 2% copper, up to a maximum of 17% copper. However, on a deposit scale, copper does not contribute significantly to the metal budget of the Roughrider Deposit.

6.5 Alteration

Strong alteration has been intersected in the Athabasca sandstone and in the highly deformed basement rocks. Alteration within the overlying Athabasca Group includes intense bleaching, limonitization, desilicification and silicification, hydrothermal hematization, and illitic argillization. None of the primary hematite in the sandstone is preserved within the zone of bleaching and alteration.

Away from the Roughrider West and East Zones, the background dominant clay species within the Athabasca sandstone is the regional dickite assemblage; within the Roughrider Deposit, it is illite. However, the extent and intensity of the alteration in the Athabasca sandstone at the Roughrider East Zone is less than that above the Roughrider West Zone. In contrast, however, the illite abundance in the sandstone above the Roughrider Far East Zone, the deepest of the three zones, is the stronger than at seen above either the East or West Zones. Consequently, this variation cannot be simply due to the deeper depth of mineralization t at the Roughrider East Zone. Currently, drilling has not identified the cause of the illite alteration patterns observed at the Roughrider Far East Zone.

In basement rocks, alteration extends to at least 180 m below the unconformity and up to 115 m laterally away from the known mineralization. It varies in strength, ranging from weak to intense where massive clay has completely replaced the protolith.

Clay alteration is predominantly white to pale green in colour and illitic in nature, and extends downward into the Archean rocks. Chlorite alteration occurs is more variable in concentration and location with respect to high grade uranium mineralization, than the Roughrider East Zone. Hematite alteration within the basement rocks is spatially restricted in distribution and is commonly associated with high-grade mineralization.

The hematite is locally variably altered to a limonitic iron oxide. Silicification is rare.

7 Deposit Types

The target mineralization on the Roughrider Project is an Athabasca unconformity-type uranium mineralization, or some variant thereof. Jefferson et al. (2007) offered the following description of the geological environment of this type of mineralization.

Fundamental aspects of the Athabasca unconformity-type uranium deposit model include reactivated basement faults and two distinct hydrothermal fluids. Reactivation of brittle basement seated faults has fractured the overlying sandstones. The faults provide a plumbing system for reducing fluids that originate in the basement. These fluids interact with a second, oxidizing fluid that originates within the Athabasca sandstone stratigraphy. The latter fluid migrates through the inherent porosity in the sedimentary rocks. In appropriate circumstances, the two fluids mix and precipitate uranium in structural traps at or near the basal Athabasca-basement unconformity. Mineralization may occur (or may have been remobilized) in the controlling fault structures well above the unconformity.

Unconformity-associated uranium deposits are pods, veins, and semi-massive replacement bodies consisting mainly of pitchblende. These deposits typically occur close to basal unconformities, in particular those between Proterozoic conglomeratic sandstone basins and metamorphosed basement rocks. Prospective basins in Canada are filled by thin, relatively flat-lying, and widely unmetamorphosed but pervasively altered, Proterozoic (approximately 1.8 Ga to less than 1.55 Ga), mainly fluvial, red bed sediments. They comprise quartzose conglomerate, sandstone, and mudstone.

The basement gneiss was intensely weathered and deeply eroded with variably preserved thicknesses of reddened, clay-altered, hematitic regolith grading down through a green chloritic zone into fresh rock. The basement rocks typically comprise highly metamorphosed Archean to Paleoproterozoic granitoid and supracrustal gneiss including graphitic metapelites that hosts many of the uranium deposits. The bulk of the uranium-lead isochron ages on pitchblende are in the range of 1600 million years (“Ma”) to 1350 Ma.

Economic amounts of pitchblende mineralization typically occurs either as monometallic, generally basement-hosted pitchblende vein fill, breccia fillings, and replacements in fault zones or as polymetallic, commonly sub-horizontal, semi-massive replacement pitchblende lenses just above or straddling the unconformity. Polymetallic mineralization contains variable amounts of uranium, nickel, cobalt and arsenic as well as traces of gold, platinum-group elements, copper, rare-earth elements and iron.

Two end-members of the deposit model have been defined (Quirt, 2003). A sandstone-hosted egress-type model (one example is the Midwest A deposit) involves the mixing of oxidizing sandstone-hosted brine with relatively reduced fluids from the basement in the sandstone. Basement-hosted, ingress-type deposits (one example is the Rabbit Lake deposit) formed by fluid-rock reactions between an oxidizing sandstone brine and the local wall rock of a basement fault zone. Both types of mineralization and associated host-rock alteration occurred at sites of basement–sandstone fluid interaction where a spatially stable redox gradient or front was present. Although either type of deposit can result in high grade pitchblende mineralization with up to twenty percent pitchblende, they are not physically large.

In plan view, the deposits can be up to 150 m long and up to thirty metres wide and/or thick. Egress-type deposits tend to be polymetallic (uranium-nickel-cobalt-copper-arsenic) and typically follow the trace of the underlying graphitic pelites and associated faults along the unconformity. Ingress-type, essentially monomineralic uranium deposits, can have more irregular geometry.

Unconformity-type uranium deposits are surrounded by extensive alteration envelopes. In the basement, these envelopes are typically narrow but become broader where they extend upwards into the Athabasca group for tens of metres to even 100 m or more above the unconformity. Hydrothermal alteration is variously marked by chloritization, tourmalinization, several episodes of hematization, illitization, silicification or desilicification, and dolomitization (Hoeve, 1984).

8 Exploration

Exploration work conducted at the Roughrider Project by Hathor includes a number of geophysical (electromagnetic, magnetic, gravity, seismic and resistivity) surveys completed by a number of different contractors between 2005 and 2009, and relogging of available historical drill core in 2006 by Hathor.

8.1 2005 GEOTEM and Aeromagnetic Survey

Fugro Airborne Surveys ("Fugro") completed a124 line kilometre airborne electromagnetic (GEOTEM) and aeromagnetic survey of the Roughrider Project area (ML-5544) in 2005 (Robertshaw, 2006). The survey did not detect any graphitic-type basement conductors within the Roughrider Project area. Three weak and short electromagnetic conductor segments, thought to represent fault zones extending through the Athabasca Group sandstone, were identified.

8.2 2006 Logging of Historic Drill Core

In the fall of 2006, Hathor relogged available historic drill core from the property. Detailed lithogeochemical and clay speciation studies of the historic drill core were also undertaken. These data were invaluable in identifying drill target areas.

8.3 2007 Aeromagnetic Survey

Goldak Airborne Surveys ("Goldak") carried out an 850 line kilometre tri-axial aeromagnetic survey in 2008. This survey provided a high quality product with sufficiently broad coverage to assess the geological and structural setting of the Roughrider Project, in relation to significant nearby features such as the uranium deposits of the adjacent MWJV owned by Areva Resources Canada Inc. (69.16%), Denison Mines (25.17%), and OURD (Canada) Co., Ltd. (5.67%). Within the Midwest Northeast, prominent structures trend 30, 50, and 95º (Robertshaw, 2008).

8.4 2007 Tempest and Magnetic Gradiometer Survey

Fugro completed a 395 line kilometre airborne electromagnetic (TEMPEST) and magnetic gradiometer survey in 2007. The survey was aimed at identifying sandstone alteration features using an early time electromagnetic channel data.

Results showed a 1 km wide region of early channel conductivity that coincided with a group of anomalies from ground resistivity surveys, including a low resistivity zone that is interpreted to identify the hydrothermal alteration associated with the Roughrider Uranium Deposit.

Although the resolution of this survey was poor due to the near-alignment of flight lines with regional flight lines, the electromagnetic signatures imply that the east-west structural trend that traverses the deposit may incorporate graphitic metasediments, and the adjacent granitic block to the south, which was identified from aeromagnetics may be overlain partly by a veneer of graphitic metasediments (Robertshaw 2008).

8.5 2007 – 2008 Ground Gravity Surveys

In 2007 and 2008, MWH Geo-Surveys Ltd. carried out several ground gravity surveys with particular emphasis on the immediate vicinity of the Roughrider Uranium Deposit. These surveys identified numerous gravity low features, interpreted to be due to alteration effects in sandstone; the strongest of which were G1, G2 and G3.

The G1 anomaly (-0.25 milligal amplitude) is spatially associated with the Roughrider West Zone and consists of two lobes extending for over 300 m trending 50º, with the Roughrider West Zone located at a saddle between the two lobes of the G1 anomaly.

The G2 anomaly (-0.15 milligal amplitude) is centred 300 m to the east of the Roughrider West Zone, and is located on the southern margin of the Roughrider East Zone.

The G3 gravity anomaly (-0.50 milligal amplitude) is a strong and relatively narrow gravity low, located in ML-5546, about 600 m further to the southwest of G1(Robertshaw, 2008). Subsequent drilling of the G3 anomaly has not intersected any alteration within the sandstone and no elevated radioactivity in either the sandstone or basement. The reason for the G3 anomaly remains unexplained.

8.6 2007 3D Seismic Survey

In 2007, Hathor commissioned a detailed three dimensional (“3D”) seismic reflection survey at the Roughrider Project (Hajnal, 2008). Details of data acquisition and processing are provided in Keller et al (2011). A first pass qualitative interpretation of the seismic data was achieved using Gocad’s 3D imaging software. 3D analysis of the data generated a framework of prominent features, mostly oriented at 65º, which were interpreted to represent major structural components within the survey area. The interpretation will be updated and refined with the addition of results, including faults and alteration zones, as encountered in subsequent exploration drilling.

8.7 2008 VTEM Survey

Geotech Ltd. carried out a 568 line kilometre helicopter-borne time-domain electromagnetic (“VTEM”) survey at the Roughrider Project in 2008. Principal geophysical sensors included a VTEM system and a caesium magnetometer. Ancillary equipment included a GPS navigation system and a radar altimeter. A total of 568 line kilometres were flown. This VTEM survey was designed to locate bedrock conductors on the Roughrider Project, but none were found (Geotech, 2009).

8.8 2007 – 2009 Resistivity Surveys

Between 2007 and 2009, six resistivity surveys were completed at the Roughrider Project, with a combined total of 125.05 km of profile coverage. Modelling results (using the Loki code) indicate intense shallow, possibly surficial, low resistivity material in the north western region of McMahon Lake. At mid-sandstone depths (74 to 136 m) an east-west trending corridor of low resistivity has been interpreted to represent alteration features in the sandstone associated with the underlying mineralization. This interpretation includes the known mineralization of the Roughrider Uranium Deposit.

At greater depths, the model indicates a significant low resistivity zone in basement rocks associated with the Roughrider East Zone. This zone of low resistivity forms a south-southwest oriented trend, extending south-southwest from the vicinity of the Roughrider East Zone and continuing southward into the adjacent Midwest Joint Venture, where it is believed to be associated with a graphitic trend in basement rocks.

A weaker low resistivity zone in the basement extends in an east-northeast orientation from the northeast of the Roughrider East Zone and appears to terminate towards the east in the vicinity of line 2760 E at a distance of approximately 1.5 km from the Roughrider East Zone.

9 Drilling

Hathor has conducted a number of drilling programs on the Roughrider Project since 2007.

A summary of drilling completed to date since 2007 is provided in Table 9.1 and in Figure 9.1.

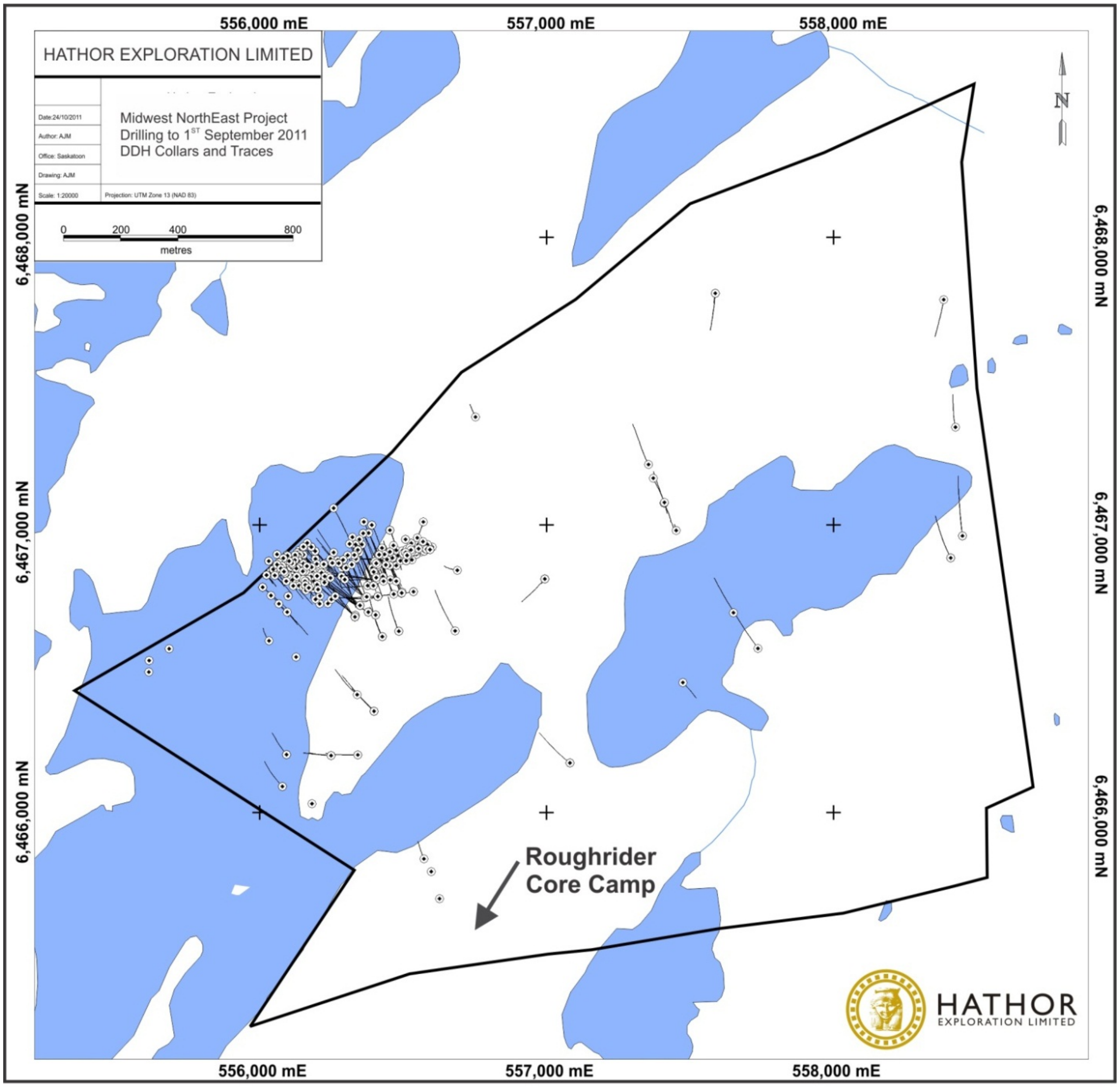


Figure 9.1: Map Showing Drilling Completed on the Roughrider Project as of September 2011

Table 9.1: Details of Diamond Drilling Programs between 2007 and 2011

RRW = Roughrider West Zone, RRE = Roughrider East Zone, RRFE = Roughrider Far East Zone, Recon = Reconnaissance

Program	Number of Holes	Drillhole Series (MWNE-)	Area	Metres Drilled	Number of Drill Rigs	Drilling Contractor
Winter 2007	3	07-01 to 07-03	Recon	906.0	1	BLY
Winter 2008	29	08-01 to 08-29	Recon, RRW	10,710.0	2	BLY
Summer 2008	13	08-30 to 08-42	RRW	5,322.2	1	TEAM
Winter 2009	89	09-43 to 09-131	RRW	31,374.4	4	TEAM
Summer 2009	57	09-132 to 09-172, 09-500 to 09-515	RRW, RRE, Recon	20,472.9	3	TEAM
Winter 2010	77	10-173 to 10-219, 10-600 to 10-629	RRW, RRE	26,925.9	4	TEAM
Summer 2010	52	10-220 to 10-242A, 10-630 to 10-658	Recon, RRW, RRE	18,840.6	3	TEAM
Winter 2011	49	11-516 to 11-523, 11-659 to 11-699	Recon, RRE, RRFE	19,234.3	3	TEAM
Summer 2011	20	11-700 to 11-719	Recon, RRFE	9,628.3	2	TEAM
Total	**389**			**143414.6**		

9.1 Drilling Not Considered for Resource Estimates

9.1.1 Roughrider West Zone

Three drillholes have been completed on the Roughrider Deposit for metallurgical reasons, MWNE-09-85, MWNE-09-171, and MWNE-09-172. Information from these drillholes were used to help constrain mineral wireframes, but was not considered for mineral resource estimation; the assay results were not available when SRK was commissioned to prepare the mineral resource model.

Furthermore, in the summer of 2010, 18 core boreholes tested the peripheral margins of the Roughrider West Zone. These data were not available when SRK was commissioned to prepare the mineral resource model and not considered for the mineral resource estimation program (Table 9.2).

Table 9.2: Composited Assay Data Highlights from Drillholes Completed at the West Zone in Summer 2010 and not Included in Current Resource Estimate

DDH	From (m)	To (m)	Grade (U_3O_8 %)	Thickness (m)	GT
MWNE-09-85	No Assays Available				
MWNE-09-171	221.5	230.0	0.25	8.5	2.15
And	238.0	241.5	0.69	3.5	2.42
MWNE-09-172	208.5	237.0	1.04	28.5	29.54
And	246.5	255.5	5.71	9.0	51.41
MWNE-10-220	237.5	245.0	3.91	7.5	29.34
MWNE-10-221	254.5	255.0	1.33	0.5	0.67
MWNE-10-226	Non Mineralized				
MWNE-10-227A	212.0	227.0	1.23	15.0	18.51
MWNE-10-228	233.0	245.5	1.58	12.5	19.75
MWNE-10-229	237.0	238.5	7.98	1.5	11.97
And	254.5	263.0	0.96	8.5	8.15
MWNE-10-230	248.0	260.0	0.24	12.0	2.84
And	269.5	272.5	1.23	3.0	3.68
MWNE-10-231	262.5	264.0	0.65	1.5	0.97
MWNE-10-232	258.0	260.0	0.05	2.0	0.10
MWNE-10-233A	253.5	256.5	0.15	3.0	0.46
MWNE-10-234	257.0	258.0	1.09	1.0	1.09
MWNE-10-235	Non Mineralized				
MWNE-10-236	Non Mineralized				
MWNE-10-237	214.0	214.5	0.16	0.5	0.08
MWNE-10-238	258.5	259.5	0.08	1.0	0.08
MWNE-10-239	245.0	246.5	0.36	1.5	0.54
MWNE-10-240	227.0	240.0	1.62	13.0	21.05
MWNE-10-241	218.5	243.5	0.98	25.0	24.59

9.1.2 Roughrider East Zone

Eleven core boreholes were completed within and immediately around the margins of the current mineralization outline of the Roughrider East Zone during the winter 2011 drilling program (Table 9.3); most of these drillholes were outside the current extents of the Roughrider East Zone. These data were not available when SRK was commissioned to prepare the mineral resource model and thus were not considered for mineral resource estimation.

Table 9.3: Composited Assay Data Highlights from East Zone Completed during Winter 2011 and not Included in Current Resource Estimate

DDH	From (m)	To (m)	Grade (U_3O_8 %)	Thickness (m)	G x T (U_3O_8% M)
MWNE-11-660D	273.5	274.5	0.44	1.0	0.44
MWNE-11-662	253.0	257.5	0.19	4.5	0.84
And	264.0	266.0	0.50	2.0	1.01
MWNE-11-664	267.5	281.0	0.60	13.5	8.16
And	295.0	298.5	1.61	3.5	5.63
MWNE-11-666	250.0	261.5	0.78	11.5	8.94
MWNE-11-668	270.0	273.5	0.86	3.5	3.03
MWNE-11-669A	269.0	270.5	0.05	1.5	0.08
MWNE-11-670	Non Mineralized				
MWNE-11-678	294.0	302.5	0.11	8.5	0.94
And	306.0	308.5	0.42	2.5	1.05
MWNE-11-680	262.5	300.5	1.96	38.0	74.45
Including	282.0	283.5	21.93	1.5	
MWNE-11-682	266.5	269.5	0.42	3.0	1.25
MWNE-11-685	281.5	285.5	0.33	4.0	1.34

9.1.3 Roughrider Far East Zone

As stated previously, a third zone, the Roughrider Far East Zone, was discovered during the winter 2011 drill program. The current outline of the Far East Zone is defined by mineralization in 28 of 40 drillholes completed in the immediate vicinity of Roughrider Far East Zone (Table 9.4; Figure 9.4); weak mineralization in other drillholes is not included in the current outline of the Far East Zone. The best intersection to date is drillhole MWNE-11-715, which intersected 7.91% U_3O_8 over a core length interval of 27.0 m. Mineralization remains open to the south and southeast.

No mineral resources are currently available for the Far East Zone and the Far East Zone was not included in this report.

Assays are pending for 2 drillholes (MWNE-11-716 and MWNE-11-718) from the Far East Zone. Drillhole MWNE-11-718 intersected a total of 17.30 m of off-scale radioactivity; the greatest amount of off-scale radioactivity within a single drillhole at Far East.

This radioactivity was intersected in two zones: one at a depth correlative with other drillhole intersections at Far East, and; one at a much shallower depth, only 45 m below the unconformity. The shallower zone presents significant potential for the discovery of an additional zone in the Roughrider system.

Table 9.4: Composited Assay Data Highlights from Far East Zone

DDH	From (m)	To (m)	Grade (U_3O_8 %)	Thickness (m)	GT
MWNE-11-661	296.0	305.5	0.76	9.5	7.22
And	392.5	396.0	0.38	3.5	1.35
MWNE-11-663	374.5	78.5	1.81	4.0	7.24
And	383.5	384.5	2.00	1.0	2.00
MWNE-11-667	327.5	365.0	1.57	37.5	58.90
Including	332.0	333.0	13.5	1.0	
MWNE-11-671A	251.0	255.0	0.53	4.0	2.10
MWNE-11-673	313.5	316.0	0.54	2.5	1.36
MWNE-11-683	332.0	338.5	0.79	6.5	5.13
And	341.0	342.0	15.46	1.0	15.46
And	345.0	372.5	0.89	27.5	24.60
MWNE-11-687	344.5	384.5	2.05	45.5	93.30
Including	358.0	360.0	22.31	2.0	
MWNE-11-692A	303.0	307.0	4.00	5.0	20.00
And	319.0	340.0	3.42	21.0	71.80
And	378.0	378.5	21.1	0.5	10.55
MWNE-11-693	336.0	387.5	1.58	48.5	76.87
MWNE-11-694	320.8	322.0	4.9	1.2	5.87
And	331.0	336.8	0.93	5.8	5.42
And	339.75	342.5	2.83	2.75	7.78
MWNE-11-695	342.5	393.5	1.69	51.0	86.20
MWNE-11-696	347.3	350.7	10.76	3.4	35.58
And	356.1	368.8	3.10	12.7	39.40
MWNE-11-698	328.0	370.8	3.26	42.8	139.50
MWNE-11-700	341.0	392.5	2.40	51.5	123.60
MWNE-11-701B	333.0	394.0	1.78	61.0	108.86
MWNE-11-702	320.5	324.0	2.74	3.5	9.58
And	338.5	350.5	2.65	12.0	31.81
And	393.5	394.0	15.70	0.5	7.85
MWNE-11-703	336.0	370.0	3.34	34.0	113.70
And	382.0	385.0	7.31	3.0	21.94
MWNE-11-704	354.5	371.5	0.78	17.0	13.28
And	378.0	383.5	1.17	5.5	6.44
MWNE-11-707	343.0	385.0	2.95	42.0	123.72
And	398.5	405.0	11.31	6.5	73.52

DDH	From (m)	To (m)	Grade (U_3O_8 %)	Thickness (m)	GT
MWNE-11-709	355.0	358.5	14.50	3.5	50.74
And	428.0	428.5	15.80	0.5	7.90
And	435.5	436.5	11.88	1.0	11.88
MWNE-11-710	332.5	390.0	1.92	57.5	110.54
MWNE-11-711	372.0	382.0	5.40	10.0	53.98
MWNE-11-712A	340.0	382.5	4.23	42.5	179.67
Including	365.5	371.5	17.09	6.0	
MWNE-11-713	406.0	410.5	5.33	4.5	23.97
MWNE-11-714	429.5	435.0	2.66	5.5	14.62
MWNE-11-715	354.5	381.5	7.91	27.0	213.57
Including	369.0	372.5	41.77	3.5	
MWNE-11-717A	384.5	386.5	2.00	2.0	4.00
And	397.5	399.0	4.35	1.5	6.52

GT – Grade x Thickness; Based on a cut-off of 0.05 % U_3O_8. All intervals are core lengths. Grade values are rounded to two decimal places. GT values have been calculated from original assay numbers which are listed to three decimal places.

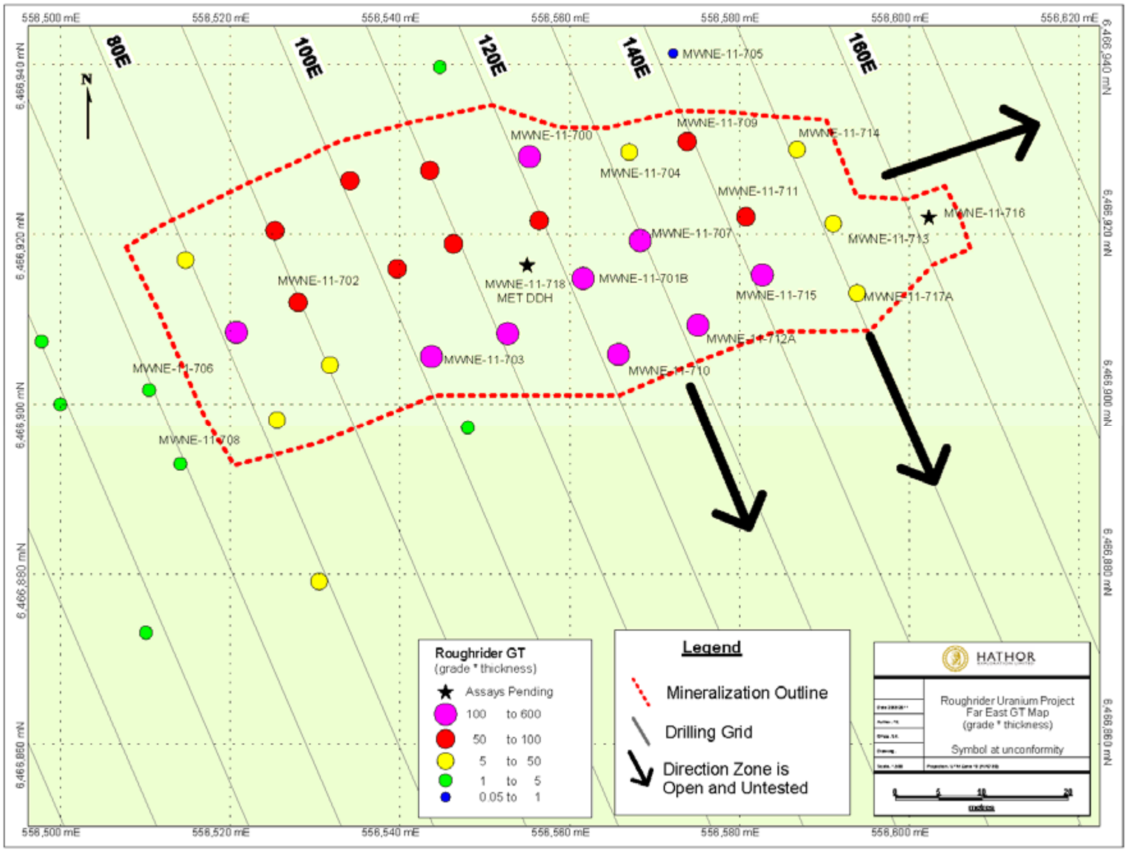


Figure 9.2: Grid line plan map of Far East Zone at Roughrider Uranium Deposit, with drillholes completed during 2011 winter (unlabelled) and summer (labelled) drill programs. Drillholes are colour-coded based on GT value (Hathor, 2011).

9.1.4 Mineralization South of Roughrider Deposit

Drillhole MWNE-11-719, collared 300 m to the southeast of the Far East Zone, intersected 1.0 m of 0.07 % U_3O_8. The surface projection of this mineralization is approximately 170 m southeast of the Far East Zone. The area between drillhole 719 and the Far East Zone remains untested and represents a high priority exploration target. This target area presents significant potential for the discovery of an additional zone at Roughrider.

9.2 Drilling Methodology

9.2.1 Roughrider West Zone

The first two diamond drilling programs on the Roughrider Project, in winter 2007 and winter 2008, were contracted to Boart Longyear Inc. in Saskatoon. Two Longyear LF-70 rigs were employed with depth capabilities of 600 m.

Since the start of 2008 summer drilling program, all drilling has been contracted to TEAM Drilling LP, from Saskatoon. These programs have utilized between one and four A5 drills, with a depth capacity in excess of 600 m with BQ, NQ and HQ rods.

Winter drill programs utilize drills mounted on metal skids to allow mobilization between drill collar sites. Summer drill programs have utilized a combination of skid-mounted, helicopter-portable and barge-based drill rigs (Figure 9.3). Both the skid-mounted and helicopter-portable rigs can complete drillholes ranging in dip from vertical to 45º. In contrast, the barge-based drill rig is limited to vertical holes. NQ-sized holes were cased NW into bedrock and drilled NQ size (47.0 ml diameter) to depth, HQ-sized holes were cased HW and drilled HQ sized (63.5 ml diameter) to depth. In rare instances, for example in hole MWNE-10-607, NQ-sized holes were reduced to BQ-sized (36.5 ml diameter) holes due to severely bad ground.

All mineralized and non-mineralized holes within the Roughrider Uranium Deposit and Roughrider East Zone systems are cemented from bottom to top. The top 30 m of all non-mineralized holes outside the Roughrider West Zone are cemented as per Saskatchewan Ministry of Environment regulations. Land-based drillhole locations are marked with a tagged picket.

With the permission of Fission Energy Corp., five drillholes were collared on their property with the intent of intersecting uranium mineralization on Hathor's claims. The drillholes are:

- MWNE-10-197;
- MWNE-10-200;
- MWNE-10-197A;
- MWNE-08-026; and
- MWNE-08-024.

All drill core located on the Fission claims was delivered to Fission Energy Corp.

Figure 9.3: Drilling Operations, Roughrider Project September, 2010. A: Barge mounted A5 drill, B: Helicopter Transported drill, C: A5 drill

9.2.2 Roughrider East Zone

All drilling on the Roughrider East Zone has been contracted to TEAM Drilling LP ("TEAM"), from Saskatoon, Saskatchewan. These programs have utilized ZMC A5 drills, with a depth capacity in excess of 600 m with BQ and NQ rods, no HQ-sized drillholes have been completed on the Roughrider East Zone.

Winter drill programs utilize drills mounted on metal skids to allow mobilization between drill collar sites. Summer drill programs utilized a combination of skid-mounted, and helicopter-portable drill. These rigs can complete drillholes ranging in dip from vertical to 45º. NQ-sized holes were cased NW into bedrock and drilled NQ size (47.0 mm diameter) to depth.

In rare instances, for example in hole MWNE-10-607, NQ-sized holes were reduced to BQ-sized (36.5 ml diameter) holes due to severely bad ground.

All mineralized and non-mineralized holes within the Roughrider East Zone were cemented from bottom to top. All land-based drillhole locations are marked with a tagged picket as per Saskatchewan Ministry of Environment regulations.

9.3 Drillhole Surveys

Holes are spotted on a grid and collar sites are surveyed by differential GPS using NAD83 and UTM Zone 13. Down hole surveys were completed either with, or a combination of, Reflex EZ-Shot or a Reflex Gyro instrument.

The Reflex EZ-shot is a single point instrument, and is used to obtain dip and azimuth measurements at 21 m intervals down the hole with an initial test taken 6 m below the casing and a final test at the bottom of the hole.

The Reflex Gryo is a continuous multi-point instrument, which is not affected by magnetics and allows measurements to be made through the casing. It is used to obtain dip and azimuth measurements at 3 m intervals through the casing and at 5 m through the rest of the hole and a final test at the bottom of the hole. The reflex Gyro system has only been employed since winter 2010.

9.4 Geophysical Surveys

At the completion of each drillhole, downhole radiometric surveys were performed down the drill string at a speed of 15 m per second down hole and 5 m per second up hole.

All holes were surveyed using a Mount Sopris winch and matrix logger interface board.

Unmineralized or weakly mineralized holes were surveyed using a single crystal (NaI) gamma probe that included the following tools: SN 3858 and 4171 for the East Zone, and SN 169, 276, 439, 3858, 4171, 4172 and 4178 for the West Zone. Holes with an estimated pitchblende content greater than 3% were surveyed with a down hole triple (one NaI and two Geiger-Mueller tubes) gamma probe that included the following tools: SN 3705 and 4484 for the East Zone, and SN 3705, 3877, 4410, and 4484 for the West Zone.

The Saskatchewan Research Council (“SRC”) provides down hole calibration test pit facilities in Saskatoon, Saskatchewan, for the calibration of down hole gamma probes. These test pits consist of four variably mineralized holes with maximum grades of 0.61%, 0.30%, 1.35%, 4.15% pitchblende.

The probes used for the surveys were calibrated at the SRC test pit facility and allow for grade thickness estimates to be made from the instrument readings and grade estimates equivalent to U_3O_8 (“*e* U_3O_8”) to be calculated.

However, it must be noted that, in general, no calibrations are available for high-grade mineralization (more than 5% U_3O_8) because Hathor has not yet been able to maintain an open, cased hole in such material and the highest grade SRC test pit available is 4.15% U_3O_8. Consequently, no *e* U_3O_8 grades are generally reported.

9.5 Drilling Pattern and Density

Diamond drilling for the Roughrider East and West Zones has been undertaken using a grid pattern with intersections for the Roughrider West Zone at approximately 10 m by 10 m and 20 m to 15 m intervals for the Roughrider East Zone. Drillholes are both vertical (inclination of 85º to 90º down) and inclined at various angles to a maximum of 45 º down.

9.6 Sampling Approach and Methodology

9.6.1 Drill Core Logging and Handling

At the drill rig, the core was removed from the core barrel by the drillers and placed directly into three row NQ wooden core boxes with standard 1.5 m length and a nominal 4.5 m capacity. Individual drill runs were identified with small wooden blocks, onto which the depth in metres was recorded. The core was transported either by the drill contractor or company personnel to Hathor’s fenced core-logging facility (Roughrider Core Camp) on the property. The core handling procedures at the drill site are industry standard.

All drill core logging and sampling was conducted by Hathor personnel. As per Hathor’s health and safety protocols, and to avoid any radioactive cross-contamination, all core boxes were scanned with a hand-held spectrometer to assess whether they were “hot” or “cold” in nature upon arrival at the Roughrider Core Camp. Hathor’s definition of “hot” core boxes are those that yield an “in-box” reading of greater than 500 cps. At this point, hot core was placed directly into the “hot shacks” and cold core (less than 500 cps) was placed in “cold shacks”.

Geologists logged the hot and cold drill core by recording their observations in a Microsoft Access-based drillhole database. The logging included observations of radioactivity, lithologies, mineralization, alteration, friability, maximum grain size in the sandstone, fracture density, structural information, core loss, and a descriptive log of the core. Upon completion of each drillhole, the data were transferred into the master database. All core trays were marked with aluminum tags as well as felt tip marker.

All mineralized core was carefully scanned with a hand-held Gamma Radiation Detector (Exploranium GR-110G or RS-120 Super SCINT) by removing each piece of drill core from the ambient background, noting the most pertinent reproducible result in cps, and carefully returning it to its correct place in the core box. These data, in conjunction with the downhole gamma probe data were used to guide split-sampling.

After selection of the intervals to be split-sampled, an aluminum Dynamo tag or a hexagonal plastic core marker with the same number was stapled into the core box at the beginning of the sample interval.

Detailed photographic records of each drillhole were kept. All drillholes were photographed from just above the marker conglomerate (approximately 160 m vertical depth below surface) to the end of the drillhole prior to sampling. Mineralized sections were additionally photographed with the sample tags in place prior to split sampling.

9.6.2 Drill Core Sampling

To determine the content and distribution of uranium, and other major, minor and trace major trace elements, as well as clay minerals (alteration), several types of samples are routinely collected from drill core from the Roughrider East and West Zones. These include:

- Composite samples of sandstone and basement rocks;
- Systematic split samples of mineralized (radioactive) drill core;
- Point samples of basement rock;
- Dry specific gravity samples; and
- Clay alteration species (PIMA) samples.

All geochemical core samples are tracked by two-part SRC ticket books. One tag goes with the sample for assay and the other tag is kept with the geologist's records.

9.6.3 Composite Samples

Hathor collected a suite of composite sandstone samples down the entire sandstone column from each drillhole. From the top of the sandstone column to a down hole depth of approximately 180 m, the sandstones were sampled by 10 m composite chip samples. For the next 20 m a total of 4 m to 5 m samples were collected, and for the final approximately 10 m up to the unconformity (approximately 210 m vertical depth below surface), 1 m to 2 m composite samples were taken. Immediately below the unconformity, a 1 m composite sample was collected from the paleoweathered material. In the case that mineralization or very strong alteration reached the sandstone column, this sampling approach was superseded by the collection of systematic split samples.

All of these composite samples were sent to the SRC laboratory for preparation and assaying.

9.6.4 Split Samples

Hathor assayed all the cored sections through mineralized intervals. Sampling of the holes for assays was guided by the radiometric logs and readings from a hand-held scintillometer. Initial drillholes (up to MWNE-08-19) were sampled using variable sample lengths between 0.2 m and 1.0 m. All drillholes after MWNE 08-19, were sampled using either 0.5 or 1.0 m sample lengths. In areas of extreme core loss sample intervals may extend locally to 3 m.

Barren samples were taken to shoulder both ends of mineralized intersections. Shoulder sample lengths were at least 1 m on either end but may be significantly more in areas with strong mineralization.

All core was split with a handheld wheel-type splitter according to sample intervals marked on the core. One half of the core was preserved in the box for future reference and the other half was bagged, tagged, and sealed in a plastic bag. The bags of samples for geochemical or clay analyses were placed in large plastic pails and sealed for shipping. Bags of mineralized samples were sealed for shipping in metal or plastic pails depending on their radioactivity. Mineralized samples were shielded by placing non-mineralized or weakly mineralized samples around the inner margins of the pail.

The mineralized rock at the Roughrider East and West Zones is predominantly highly altered basement gneisses. Locally, the core can be broken and blocky, but recovery was generally good with an estimated 90% overall recovery. Local intervals of up to 10 m with only 80% recovery have been encountered. Intervals where core loss was greater than 50% over 3 m runs were rare.

The split sample material sent for assay was regarded to accurately represent the entire core and should be free of bias because of the relatively competent nature of the core recovered.

Due to the high rate of core recovery within the mineralized zone, chemical assays are considered reliable. In rare cases, some mineralization may have washed out during the drilling process. In these instances, close correlation of the down hole gamma probe and the observed chemical analyses was undertaken. In such instances, a more accurate measurement of the pitchblende content should be determined by the gamma logging probe which was run in every hole.

9.6.5 Point Samples

Point samples, normally 10 to 15 cm in length, are taken: selectively through the paleo-weathering profile; systematically at 3 m or 5 m intervals through altered basement rock which is not split-sampled; and selectively through fresh basement rock. This sampling aids in the identification and understanding of cryptic metal distribution.

9.6.6 PIMA Sampling

For the determination of clay alteration species in the sandstone column, Hathor collects samples for analyses using the PIMA analyzer. Throughout the sandstone section, a 2 cm to 3 cm chip sample of core is collected every 5 m or 10 m. Near the unconformity, the sample interval is shortened as needed. PIMA samples are also collected as needed throughout the altered basement rocks, normally at 3 m or 5 m intervals.

10 Sample Preparation, Analyses, and Security

Drill core from the Roughrider Project was logged, marked for sampling, split, bagged, and sealed for shipment by Hathor personnel at their fenced core-logging facility on the property. All samples for pitchblende assay were transported by land, in compliance with pertinent federal and provincial regulations by Hathor personnel. The sample containers were transported directly to the Geoanalytical Laboratories of the SRC located in Saskatoon.

Non-mineralized samples for routine geochemical investigation were shipped to the Geoanalytical Laboratories of the SRC by ground transport. Samples for PIMA clay analyses were shipped to a consultant, Mr. Ken Wasyliuk of Northwind Resources Ltd., Saskatoon, by ground transport.

The Geoanalytical Laboratories of the SRC are unique facilities offering high quality analytical services to the exploration industry. The laboratory is accredited ISO/IEC 17025 by the Standards Council of Canada for certain testing procedures including those used to assay samples submitted by Hathor. The laboratory is licensed by the Canadian Nuclear Safety Commission (“CNSC”) for possession, transfer, import, export, use and storage of designated nuclear substances by CNSC Licence Number 01784-1-09.3. As such, the laboratory is closely monitored and inspected by the CNSC for compliance.

SRC is an independent laboratory, and no associate, employee, officer or director of Hathor is, or ever has been involved in any aspect of sample preparation or analysis on samples from the Roughrider East and West Zones.

Analytical data results were sent electronically to Hathor. These results were provided as a series of Adobe PDF files containing the official analytical results and a Microsoft Excel spreadsheet file containing only the analytical results. Upon receipt of the data by Hathor, the electronic data were imported directly into the Access-based master drillhole database. During the import process, all values reported below detection limits were converted to half the detection limit of that element. Hard copies of the assay certificate were mailed to Hathor’s exploration office in Saskatoon.

10.1 Sample Preparation and Analyses

10.1.1 Core Drilling Sampling

SRC performs the following sample preparation procedures on all samples submitted to them. There is no sample preparation involved for the samples sent for clay analyses.

On arrival at SRC, samples were sorted into their matrix types (sandstone or basement rock) and according to radioactivity level. The samples were prepared and analyzed in that order.

Sample preparation (drying, crushing, and grinding) was done in separate facilities for sandstone and basement samples to reduce the probability of sample cross-contamination. Crushing and grinding of radioactive samples yielding more than 2,000 cps was done in another separate CNSC-licensed radioactive sample preparation facility. Radioactive material was kept in a CNSC-licensed concrete bunker until it could be transported by certified employees to the radioactive sample preparation facility.

Sample drying was carried out at 80ºC with the samples in their original bags in large low temperature ovens. Following drying, the samples were crushed to 60% passing 2 ml using a steel jaw crusher. A 100 to 200 gram split was taken of the crushed material using a riffle splitter. This split was then ground to 90% passing 150 mesh using a chromium-steel puck-and-ring grinding mill for mineralized samples or a motorized agate mortar and pestle grinding mill for all non-mineralized samples. The resulting pulp was transferred to a clear plastic snap-top vial with the sample number labelled on the top. All grinding mills were cleaned between sample runs using steel wool and compressed air. Between-sample grinds of silica sand were performed in case the samples were clay-rich.

Prior to the primary geochemical analysis the sample material was digested into solution using several digestion methods. A “total” three-acid digestion on a 250 ml aliquot of the sample pulp uses a mixture of concentrated $HF/HNO_3/HClO4$ acids to dissolve the pulp in a Teflon beaker over a hotplate; the residue, following drying, was dissolved in 15 ml of dilute ultrapure HNO_3. A “partial” acid digestion, on a two gram aliquot of the sample pulp, was digested using 2.25 ml of an eight to one ratio of ultrapure HNO_3 and HCl for one hour at 95ºC in a hot water bath and then diluted to 15 ml using deionized water.

For fluorimetric analysis of uranium, an aliquot of either total digestion solution or partial digestion solution was pipetted into a platinum dish and evaporated. A NaF/LiK pellet was placed on the dish and the sample was fused for three minutes using a propane rotary burner and then cooled to room temperature before fluorimetric analysis.

Another digestion method used was a sodium peroxide fusion in which an aliquot of pulp was fused with a mixture of Na_2O_2 and $NaCO_3$ in a muffle oven. The fused mixture was subsequently dissolved in deionised water. Boron was analyzed by inductively coupled plasma optical emission spectrometry on this solution.

With each batch of samples run, SRC inserts, at a minimum, a duplicate from the batch and a quality control standard of its own. For analytical quality control purposes, Hathor inserted one field duplicate for approximately every 10 m of sampled interval. This frequency equates to one duplicate for every twenty samples. Prior to Winter 2010, all field duplicates were quarter core in size, and since winter 2010 all field duplicates were half core in size.

Hathor also instructed SRC to run one coarse reject duplicate with every batch of 20 samples. Furthermore, in 2010 Hathor submitted one blank sample per drillhole.

After standard sample preparation, SRC analyzed the samples by several analytical methods depending on the characteristics of each sample:

- All split samples, both mineralized and non-mineralized, from within the mineralized section were assayed for pitchblende using SRC accredited U_3O_8-method (code U_3O_8).
- All split samples were additionally analyzed using inductively coupled plasma optical emission spectrometry (“ICP-OES”) (partial and total digestion; method code ICP-1), plus boron;
- Select split samples were analyzed for gold, platinum, and palladium by conventional fire assay procedures and axial inductively coupled plasma spectrometry finish on fifteen gram sub-samples (method code AU5); and
- Non-radioactive, non-mineralized samples were analyzed using ICP-OES (partial and total digestion; method code ICP-1) and/or inductively coupled plasma mass spectrometry (“ICP-MS”) (partial and total digestion; method code ICPMS 1), plus boron.

10.2 Specific Gravity Data

10.2.1 Roughrider West Zone

In winter 2009 (MWNE 09-43A onwards), Hathor instituted a process to determine the dry specific gravity on un-split core samples from various host rocks and mineralization styles. These samples were dried for a minimum of two to three days. Dry specific gravity was determined by the water immersion methodology. Dried core pieces were weighed, wrapped in plastic film which was heated to make tight seal around the core, and then weighed suspended in water. Dry specific gravity was determined for 50 cm core lengths to correspond to the sample interval. Prior to the determination of specific gravity of the unknown samples, specific gravity is determined on three in-house standards. The specific density database used in the West Zone resource estimation contains 409 dry specific gravity determinations, including 366 measurements on core samplesfrom the Roughrider West Zone.

In addition, twelve samples from drillholes prior to 09-43A were submitted to Geoanalytical Laboratories of SRC for dry specific gravity determination. Hathor also submitted twenty-six samples to the SRC Geoanalytical Laboratories to check the specific gravity measured onsite.

10.2.2 Roughrider East Zone

The specific density database used in the East Zone resource estimation includes 89 dry specific gravity determinations, including 46 measurements on core samples from the Roughrider East and West Zone high grade mineralization envelope.

Six core specific gravity samples from the Roughrider East and West Zones were submitted to an umpire laboratory at SRC.

10.3 Quality Assurance and Quality Control Programs

SRK has not reviewed the quality control data for the Far East Zone.

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. This includes written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying.

They are also important to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and insertion of quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process.

Check assaying is typically performed as an additional reliability test of assaying results. This typically involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

Hathor relied partly on the internal analytical quality control measures implemented by SRC. In addition, Hathor implemented external analytical quality control measures consisting of using control samples in all sample batches submitted for assaying and requesting replicate pulp assays for every 20th sample. External quality control checks consisted of inserting a field duplicate approximately every ten metres of sampling and a field blank approximately every 10 m.

For the Roughrider East and West Zones, SRC used five control samples including five standards prepared by CANMET of Natural Resources Canada (BL2A, BL3, BL4A, BL5, UHU-1 and CUP2).

Umpire assays were sent to SRC Analytical Laboratory and analyzed by a separate laboratory using Delayed Neutron Counting (“DNC”) for uranium analysis.

11 Data Verification

11.1 Verifications by Hathor

As outlined in Section 10.3, Hathor relied partly on the internal analytical quality control measures implemented by SRC and also implemented external analytical quality control measures consisting of control samples in all sample batches submitted for assaying.

During drilling, experienced Hathor geologists implement practical measures designed to ensure the reliability and trustworthiness of exploration data acquired on the Roughrider Project. In the opinion of SRK, the field procedures used by Hathor generally meets or exceeds “industry best practices”.

Sample shipments and assay deliveries were routinely monitored as produced by the primary laboratory.

11.2 Verifications by SRK

11.2.1 Site Visit

Roughrider West Zone

In accordance with the National Instrument 43-101 guidelines, G. David Keller from SRK visited the Roughrider Project during the period of September 13 to 14, 2010 accompanied by Alistair McCready representing Hathor. During the time of the site visit drilling operations were ongoing. The purpose of the site visit was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property and assess logistical aspects and other constraints relating to conducting exploration work in this area.

All aspects that could materially impact the mineral resource evaluation reported herein were reviewed with Hathor staff. SRK was given full access to all relevant project data. SRK was able to interview exploration staff to ascertain exploration procedures and protocols.

Drillhole collars are clearly marked with stakes inscribed with the borehole number on an aluminum Dynamo labels. No discrepancies were found between the location, numbering or orientation of the holes verified in the field and on plans and the database examined by SRK.

During the site visit SRK examined and briefly relogged mineralized zones for eighteen boreholes within the envelope of mineral resources (Table 11.1).

Table 11.1: Boreholes Selectively Relogged by SRK

Drillhole ID		
MWNE-10-209	MWNE-09-094	MWNE-09-161C
MWNE-10-197A	MWNE-09-097	MWNE-08-040
MWNE-09-129	MWNE-09-101	MWNE-09-056
MWNE-10-200	MWNE-10-191	MWNE-09-131
MWNE-09-116	MWNE-10-192	MWNE-09-079A
MWNE-08-226	MWNE-10-182	MWNE-09-110C.

Roughrider East Zone

In accordance with the NI 43-101 guidelines, G. David Keller visited the Roughrider Project during the period of March 16 to 18, 2011 accompanied by Alistair J. McCready, Vice President Exploration and Tom Elash, Project Geologist representing Hathor. During the time of the site visit drilling operations were ongoing. The purpose of the site visit was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property and assess logistical aspects and other constraints relating to conducting exploration work in this area.

All aspects that could materially impact the mineral resource evaluation reported herein were reviewed with Hathor staff. SRK was given full access to all relevant project data. SRK was able to interview exploration staff to ascertain exploration procedures and protocols.

Drillhole collars are clearly marked with stakes inscribed with the borehole number on an aluminum Dynamo labels. No discrepancies were found between the location, numbering or orientation of the holes verified in the field and on plans and the database examined by SRK.

During the site visit SRK examined and briefly relogged mineralized zones for sixteen boreholes within the envelope of mineral resources (Table 11.2). SRK also briefly relogged and examined five drillholes from the Roughrider Far East Zone (MWNE-11-667, MWNE-11-687, MWNE-692A, MWNE-11-693, and MWNE-11-694).

Table 11.2: Boreholes Selectively Reviewed by SRK

Drillhole ID	
MWNE-09-170	MWNE-10-626A
MWNE-10-602A	MWNE-10-630A
MWNE-10-607	MWNE-10-630A
MWNE-10-609	MWNE-10-648
MWNE-10-610	MWNE-10-649
MWNE-10-613	MWNE-10-656A
MWNE-10-615	MWNE-11-664
MWNE-10-624B	MWNE-11-666

11.2.2 Database Verification

SRK conducted a series of routine verifications to ensure the reliability of the electronic data provided by Hathor. These verifications include auditing the electronic data against original records for the Roughrider East and West Zones. Approximately 10% of the assay data were audited for accuracy against assay certificates received directly from Hathor and approximately 5% from the primary laboratory (“SRC”). No data entry errors were noted. In the opinion of SRK, the electronic data are reliable and free of material data entry errors.

11.2.3 Verifications of Analytical Quality Control Data

SRK has not completed a review of the Roughrider Far East Zone Quality control data at the time of writing this report.

Roughrider West Zone

Hathor made available to SRK analytical quality control data in the form of a Microsoft Excel spreadsheet that contained the assay results for the quality control samples inserted with samples submitted for assaying. SRK aggregated the assay results for the external quality control samples for further analyses. Sample blanks and internal SRC standards data were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates and umpire check assays) were analyzed using bias charts, quantile-quantile and relative precision plots. The analytical quality control data produced by Hathor between 2008 and 2010 are summarized.

The internal and external quality control data produced on this project represents approximately forty percent of the total number of samples submitted for assaying (Table 11.3). Discounting the laboratory inserted internal quality control points, this percentage decreases to 24%. This ratio is excellent and exceeds industry best practice.

In general, the performance of the control samples inserted with samples submitted for assaying is generally acceptable. Inspection of time series for field blank samples indicates that the primary laboratory performed well.

Table 11.3: Summary of Hathor Analytical Quality Control Data

	West	East	Total	(%)	Comment
Assay Sample Count	3,956	3,353	**7,309**		
Field Blanks	111	76	**187**	**2.56**	
Standards	0	0	**0**	**0.00**	
Field Duplicates	393	163	**556**	**7.61**	
Total External QC Samples	504	239	**743**	**10.17**	
Blanks	0	0	**0**	**0.00**	
Standards	875	453	**1,328**	**18.17**	
BLA-2a	11	1	**12**		CANMET (0.502% U3O8)
BL-3	83	45	**128**		CANMET (1.21% U3O8)
BL-4a	633	333	**966**		CANMET (0.147% U3O8)
BL-5	88	49	**137**		CANMET (8.36% U3O8)
UHU-1	3	0	**3**		SRC (80.5% U3O8)
CUP-2	57	25	**82**		CANMET (87.5% U3O8)
Pulp Duplicates	418	157	**575**	**7.87**	
Reject Duplicates	484	159	**643**	**8.80**	
Total Internal QC Samples	1777	769	**2,546**	**34.83**	
Check Assays					
SRC Geoanalyical and SRC Analytical	938	585	**1,523**	**20.84**	
Total Assay QC Samples	**3,219**	**1,593**	**4,812**	**65.84**	
Density Sample Count	421	89	**510**		
Standards	147	63	**210**	**41**	
Std 01	48	21	**69**		Hathor
Std 02	49	21	**70**		Hathor
Std 03	50	21	**71**		Hathor
Field Duplicates	26	6	**32**	**6.3**	
Total Density QC Samples	**173**	**69**	**242**	**47**	

Roughrider East Zone

Hathor made available to SRK analytical quality control data in the form of a Microsoft Excel spreadsheet that contained the assay results for the quality control samples. Sample blanks and internal SRC standards data were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates of core and laboratory duplicates of pulp, pulp rejects and umpire laboratory checks samples) were analyzed using bias charts, quantile-quantile and relative precision plots. The analytical quality control data produced by Hathor between 2009 and 2010 for the Roughrider East Zone are summarized in Table 11.3. The internal and external quality control data produced on this project represents approximately 30% of the total number of samples submitted for assaying (Table 11.3). Discounting the laboratory inserted internal quality control points, this percentage decreases to approximately 7%. This ratio is excellent and exceeds industry best practice.

In general, the performance of the control samples inserted with samples submitted for assaying is generally acceptable. Inspection of time series for field blank samples indicates that the primary laboratory performed well.

12 Mineral Processing and Metallurgical Testing

This section summarizes Phase I and Phase II metallurgical testwork completed on the Roughrider mineralization at SGS Canada Inc.-Lakefield Research ("SGS Lakefield") under the direction of Melis Engineering Ltd. ("Melis").

Metallurgical testing has not been completed on representative samples from the Roughrider East Zone. Testing was, however, conducted on representative uranium mineralization samples from the nearby Roughrider West Zone, the characteristics of which are similar to Roughrider East Zone. SRK therefore believes that it is relevant to present the testing results obtained from Roughrider West Zone.

12.1 Summary

Test work on Roughrider test composites at SGS Lakefield under the direction of Melis was initiated in late 2008 and continued into 2009 and 2010 as part of development work on Hathor's Roughrider Uranium Project located in northern Saskatchewan. The first two phases of test work completed on Roughrider West Zone mineralization, reported in this section and used for the process component of the Roughrider PEA, included preparation of variability and overall composites, comminution tests, acid leaching tests, uranium recovery and precipitation tests, tailings preparation, effluent treatment and environmental analyses.

While individual Roughrider assay samples may contain localized high concentrations of base metals, analysis of test composites showed that, on a mining-scale, the Roughrider mineralization contains relatively low levels of associated elements such as arsenic, selenium and base metals.

SAG mill power indices and ball mill Bond Work Indices measured on variability composites yielded an average SAG mill power index of 14.3 minutes and an average ball mill Bond Work Index of 9.3 kWh/t, both indicating that the Roughrider mineralization is softer than average.

Atmospheric and low pressure leach tests identified that leach extractions of 99% or better can be obtained on an overall Roughrider test composite with a head grade of 3.30 % U_3O_8. Additional tests on other variability composites showed that the Roughrider mineralization is easily leached. Typical leach conditions are a mesh-of-grind K_{80} of 100 µm to 125 µm, a leach temperature of 50 °C, a leach retention time of 12 hours or less, a free acid of 15-20 g H_2SO_4/L achieved with an acid consumption of approximately 125 kg H_2SO_4/tonne, and an oxidation reduction potential of 500 mV or greater achieved with an average oxidant consumption of approximately 4 kg $NaClO_3$/tonne.

Uranium precipitation tests showed that yellowcake produced from Roughrider mineralization can meet typical product quality standards such as the standards for the ConverDyn uranium refinery of Englewood, Colorado.

The concentrations of all elements in the treated effluent are below typical regulatory limits set by the provincial and federal governments for Saskatchewan uranium mines. However, the regulations are moving away from assay limits to limits based on environmental loading to the local watershed, which combines assays and volume discharged. Applicable discharge limits for the Roughrider Project will be identified during the formal environmental approval process.

12.2 Metallurgical Testwork-Phase I

12.2.1 Summary

Testwork on Roughrider test composites made up from assay rejects taken from drillholes in the West Zone of the Roughrider deposit was initiated in late 2008 at SGS Lakefield under the direction of Melis as part of development work on Hathor's Roughrider Uranium Project located on the eastern side of the Athabasca Basin in northern Saskatchewan. This preliminary test program, referenced as Phase I, encompassed composite analyses, leach scoping tests, uranium recovery and upgrading, effluent treatment, and preparation of tailings to provide preliminary environmental data.

While individual Roughrider assay samples may contain localized high concentrations of base metals, analysis of the Phase I test composites, as summarized in Table 12.1 below, showed that, on a mining-scale, the Roughrider mineralization contains relatively low levels of associated elements such as arsenic, selenium and base metals.

Table 12.1: Roughrider Phase I Test Composites –Analysis of Key Elements

Analyte	Unit	Composite No. 1	Composite No. 2	Composite No. 3
U_3O_8	%	5.96	2.71	0.64
As	%	0.052	0.15	0.0065
Co	%	0.022	0.021	0.0036
Cu	%	0.077	0.12	0.042
Mo	%	0.24	0.071	0.17
Ni	%	0.025	0.066	0.0078
Pb	%	1.98	0.085	0.045
Se	%	0.0029	0.0016	<0.0001
Zn	%	0.046	0.018	<0.004

Leaching tests on the three composites grading 5.96% U_3O_8, 2.71% U_3O_8 and 0.64% U_3O_8 yielded high uranium extractions of 99.2%, 98.8% and 98.4% respectively. Optimum leach conditions were typical of northern Saskatchewan uranium ores, namely a leach temperature of 50 °C, a 12 hour leach retention time, a free sulphuric acid concentration of 20 g H_2SO_4/L and an oxidation potential of approximately 500 mV achieved with a sodium chlorate oxidant addition of 1 to 5 kg $NaClO_3$/t.

Analysis of preliminary uranium products from the test program showed that yellowcake produced from Roughrider mineralization, with appropriate circuit adjustments, can meet typical refinery quality specifications.

Tailings and effluent treatment tests showed that effluent treatment can be effective, requiring chemical dosages typically used in northern Saskatchewan uranium mines, particularly for effective removal of molybdenum.

From this first phase of preliminary work, it was concluded that uranium from the Roughrider mineralization can be recovered under relatively mild leach conditions producing a product of acceptable quality using typical northern Saskatchewan uranium process conditions.

12.2.2 Phase I Composite Preparation

Since the structure and lithology of the deposit remained to be studied at the time of sample selection, it was decided that initial test composites be prepared to reflect varying potential run-of-mine grades. Three target grades, 5% U_3O_8, 2.5% U_3O_8 and 0.5 % U_3O_8, were used to prepare test composites representing the mineralized zones, for reference purposes the three composites were named Composite Nos. 1, 2 and 3, respectively.

The test composites were made up on a weighted basis according to drillhole intersections using coarse assay rejects stored at the Saskatchewan Research Council (“SRC”) in Saskatoon, Saskatchewan. The composites were prepared by SRC as specified by Hathor, and shipped to SGS Lakefield for testing. Composite No. 1 was comprised of 42 consecutive samples from DDH 08-12 and 23 consecutive samples from DDH 08-33 for an aggregate sample length of 24.1 m. Composite No. 2 was comprised of 19 consecutive samples from DDH 08-28 and 23 consecutive samples from DDH 08-30 for an aggregate sample length of 22 m. Composite No. 3 was comprised of 21 consecutive samples from DDH 08-24 and 40 consecutive samples from DDH 08-32 for an aggregate length of 31.5 m. The majority of the sample lengths from the subject holes were 0.5 m.

Uranium analyses of the three West Zone test composites used in the Phase I test program are summarized in Table 12.2 below, and Table 12.3 summarizes the key elemental analyses of the three composites. Assays of two different samples yielded the average uranium head grade shown. The average calculated uranium head grade are the average head grades calculated from the products of leach tests (sometimes referred to as metallurgical calculated head grade). The drill indicated calculated head grade shown in the table below are weighted averages using the listed uranium grades taken from the Hathor drill core assay data.

Table 12.2: Roughrider West Zone Phase I Test Composites Head Assays

Sample	Unit	Composite No. 1	Composite No. 2	Composite No. 3
Sample No. 1	%	6.11	2.68	0.62
Sample No. 2 Re-Assay	%	5.80	2.74	0.65
Average Composite Assay Head Grade	%	5.96	2.71	0.64
Average Calculated Head Grade	%	6.17	2.60	0.57
Drill Indicated Calculated Head Grade	%	5.03	2.49	0.50

Table 12.3: Roughrider West Zone Phase I Test Composites Analyses of Key Elements

Analyte, %	Unit	Composite No. 1	Composite No. 2	Composite No. 3
U_3O_8	%	5.96	2.71	0.64
As	%	0.052	0.15	0.0065
Co	%	0.022	0.021	0.0036
Cu	%	0.077	0.12	0.042
Mo	%	0.24	0.071	0.17
Ni	%	0.025	0.066	0.0078
Pb	%	1.98	0.085	0.045
Se	%	0.0029	0.0016	<0.0001
Zn	%	0.046	0.018	<0.004
Au	g/t	1.05	0.23	0.48
Ag	g/t	34	3.1	12
ABA - NP/AP(1)	-	1.47	0.95	1.68

Notes: 1. Acid Base Accounting Measurement (Neutralization Potential/Acid Generating Potential)

The uranium analyses of the Roughrider West Zone composites prepared for testing were close to the target grades of 5.0% U_3O_8, 2.5% U_3O_8 and 0.5% U_3O_8, and the composites were relatively low in deleterious elements such as arsenic, selenium, and base metals. Molybdenum levels were somewhat elevated, which implies that downstream uranium separation and effluent treatment will need to take this into consideration when identifying treatment conditions.

The ABA (Acid Base Accounting) measurements suggest that the Roughrider mineralization is either PAG (Potential Acid Generating) or Uncertain with respect to acid generation based on the ABA measurements.

12.2.3 Scoping Leach Tests

Scoping leach tests on the three Phase I test composites included both atmospheric acid leach tests and low pressure acid leach tests, under typical conditions used in northern Saskatchewan. Extractions obtained in the scoping leach tests are shown in Figure 12.1 below.

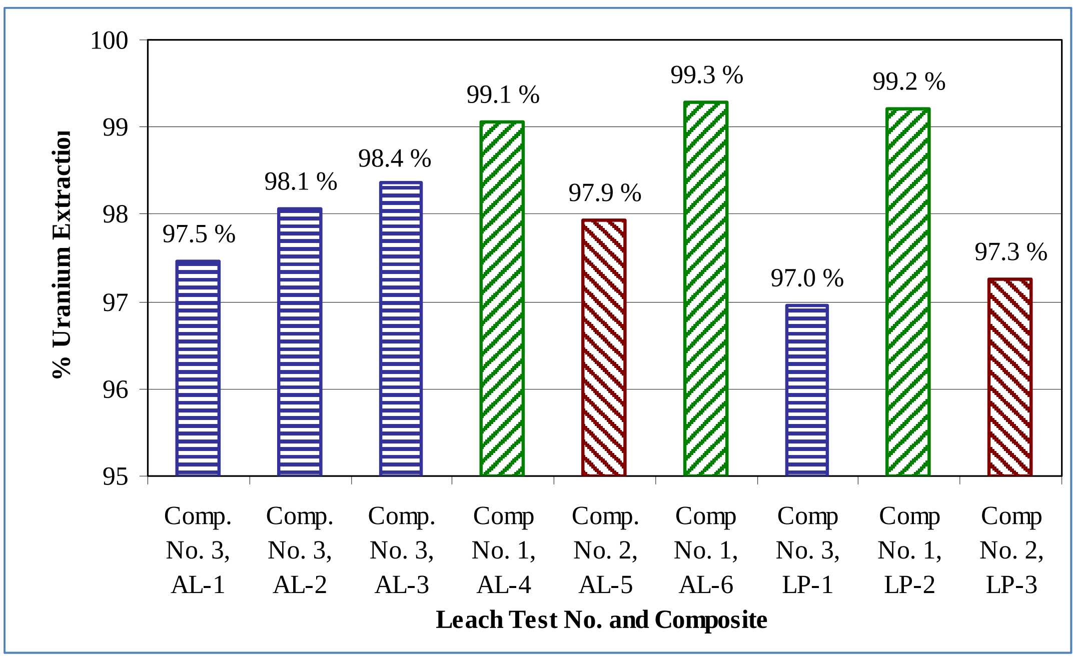


Figure 12.1: Leach Extractions in Scoping Tests on West Zone Composites

The highest extraction obtained for Composite No. 1, 99.2%, was obtained in the low pressure leach test LP-2. The highest extraction obtained for Composite No. 2, 98.8%, was obtained in the atmospheric pressure leach test AL-5. The highest extraction obtained for Composite No. 3, 98.4%, was obtained in the atmospheric pressure leach test AL-3.

The optimum leach conditions were identified as: a grind K_{80} (80% passing size) of 150 µm, a temperature of approximately 50 ºC, a free acid concentration of 20 g H_2SO_4/L, an ORP (oxidation-reduction potential) of 475 to 500 mV. These leach conditions resulted in leach extractions of greater than 98% after 24 hours in Composite Nos. 1, 2 and 3.

The minimum leaching times, defined as the leaching time after which the slope of the leach kinetics curves are close to flat, vary between composites and atmospheric and low pressure leach conditions. A shorter leach time of 12 hours in atmospheric leaching results in minimum leach extractions of 97.7%; and minimum leach extractions of 96.4% in low pressure leaching.

The sulphuric acid consumption required to obtain an average test free acid concentration of 20 g H_2SO_4/L was 57 kg/t for Composite No. 3 (containing 0.64% U_3O_8), 105 kg/t for Composite No. 2 (containing 2.71% U_3O_8) and 135 kg/t for Composite No. 1 (containing 5.96% U_3O_8).

The oxidant (sodium chlorate, $NaClO_3$) consumption required to obtain an average test ORP of approximately 475 to 500 mV was 1.2 kg/t for Composite No. 3, 4.1 kg/t for Composite No. 2 and 5.2 kg/t for Composite No. 1.

Atmospheric pressure leaching performed better on Composite Nos. 2 and 3 than low pressure leaching. Atmospheric and low pressure leaching performed about equally on Composite No. 3, hence atmospheric leaching is the preferred leach option.

12.2.4 Bulk Leach Test

A bulk leach was completed on Composite No. 3 to provide leach residue for settling tests and tailings production, and pregnant leach solution for solvent extraction tests to generate uranium rich liquor for precipitation tests and raffinate for effluent treatment tests. Analyses of the pregnant leach solution from the bulk leach test are summarized in Table 12.4 below.

Table 12.4: Roughrider West Zone Phase I Bulk Leach Test Pregnant Leach Solution Assays

Analyte	Unit	Assay
U_3O_8	g/L	2.36
As	g/L	0.025
Fe	g/L	0.32
Mo	g/L	0.16
Si	g/L	0.36
Ag	g/L	< 0.0002
Al	g/L	0.77
Ba	g/L	0.0002
Be	g/L	0.0002
Bi	g/L	0.008
Ca	g/L	0.49
Cd	g/L	0.0005
Co	g/L	0.011
Cr	g/L	0.004
Cu	g/L	0.027
K	g/L	0.47
Li	g/L	0.004
Mg	g/L	0.30
Mn	g/L	0.005
Na	g/L	0.096
Ni	g/L	0.011
P	g/L	0.013
Pb	g/L	0.014
Sb	g/L	< 0.001
Se	g/L	< 0.003
Sn	g/L	< 0.002
Sr	g/L	0.007
Ti	g/L	0.001
Tl	g/L	< 0.003
V	g/L	0.038
W	g/L	< 0.002
Y	g/L	0.02
Zn	g/L	0.035

12.2.5 Solvent Extraction

Solvent extraction tests were carried out on the pregnant leach solution prepared from Composite No. 3 using two flowsheets, one flowsheet testing an ammonium sulphate strip solvent extraction circuit and a second one testing a strong sulphuric acid strip circuit.

In the ammonium sulphate strip solvent extraction circuit, the highest extraction achieved was for uranium, at 99.95%. The second highest extraction was found with molybdenum, at 98.7%. Other impurities with relatively high extractions (>60%) were silver, arsenic, barium, bismuth, phosphorus, lead, strontium, titanium and vanadium.

In the strong sulphuric acid strip solvent extraction circuit the highest extraction achieved was for uranium, at 99.95%. The second highest extraction was found with molybdenum, at 98.1%. Other impurities with relatively high extractions (>60%) were lead and strontium.

The available metal extractions (not all extractions were calculable) for the strong sulphuric acid and ammonium sulphate flowsheets are compared below in Figure 12.2 below.

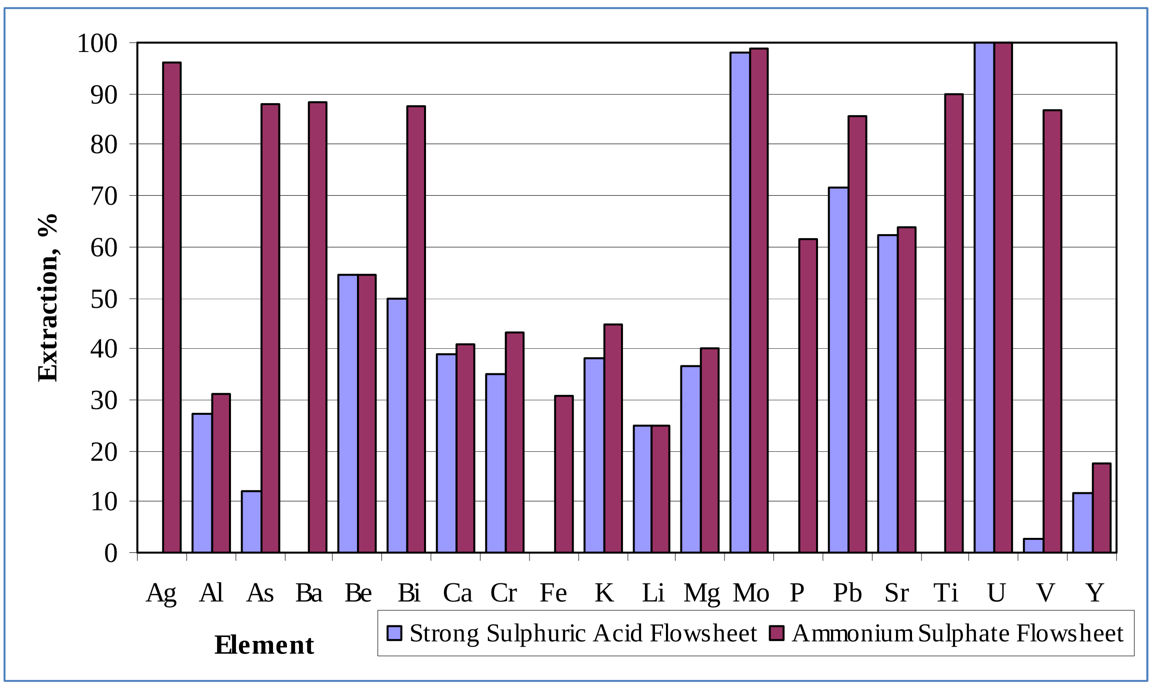


Figure 12.2: Comparison of Metal Extractions

The comparison of metal extractions makes it clear that the strong sulphuric acid flowsheet transfers fewer metals onto the organic than does the ammonium sulphate flowsheet.

Extraction isotherms for both flowsheets were relatively straight lines, and indicate that four stages would be required to reduce the uranium concentration in the pregnant leach solution to <1 mg U_3O_8/L. Under batch test conditions, no effort was made to clean the extracted molybdenum from either the organic or the loaded strip solution, as would be done in a continuous operating situation.

12.2.6 Uranium Precipitation

Uranium was precipitated from solution using two flowsheets. In the first, uranium was precipitated as ammonium diuranate and in the second, uranium was precipitated as uranyl peroxide. To judge the efficiency of the precipitation reactions, the concentration of elements in the loaded strip solutions per unit uranium were compared to their concentrations in the yellowcake precipitate as listed in Table 12.5.

Table 12.5: Comparison of Impurity Concentrations in Loaded Strip and Yellowcake Precipitate

Element	Unit	Ammonium Sulphate Flowsheet		Strong Sulphuric Acid Flowsheet	
		Loaded Strip	Ammonium Diuranate $[(NH_4)_2U_2O_7]$ Precipitate	Loaded Strip	Uranyl Peroxide $(UO_4 \bullet nH_2O)$ Precipitate
Ag	% / % U	< 0.13	<0.078	< 0.07	<0.062
As	% / % U	0.067	0.064	0.28	<0.046
Ba	% / % U	0.0044	<0.0003	0.0023	<0.0003
Ca	% / % U	0.78	0.023	1.03	0.18
Cd	% / % U	0.007	<0.002	0.009	<0.002
Cr	% / % U	0.004	0.002	0.009	<0.003
Cu	% / % U	0.24	<0.062	0.25	<0.077
Fe	% / % U	0.53	0.19	1.26	0.86
K	% / % U	0.78	<0.062	1.02	<0.12
Mg	% / % U	0.49	<0.047	0.67	1.17
Mo	% / % U	1.13	0.50	2.41	0.17
Na	% / % U	7.33	0.030	10.1	0.025
P	% / % U	0.44	<0.031	0.23	0.097
Pb	% / % U	0.18	<0.031	0.18	<0.031
Se	% / % U	0.067	<0.005	0.034	<0.005
Ti	% / % U	0.089	<0.031	0.046	<0.046
V	% / % U	0.007	<0.005	0.098	0.043

Of the assayed elements, the only impurity which increased in concentration relative to the uranium concentration was magnesium, in the strong sulphuric acid flowsheet, because magnesium hydroxide (as a 5% slurry of MgO) was used to control pH during the precipitation. Evidently, not all the MgO in the slurry converted to the soluble $Mg(OH)_2$.

A comparison of the uranium precipitate assays to the “yellowcake” specifications from ConverDyn is presented in Table 12.6.

Table 12.6: Yellowcake Precipitate Analysis

Analyte	Units	Uranyl Peroxide ($UO_4 \cdot nH_2O$)	Ammonium Diuranate [$(NH_4)_2U_2O_7$]	ConverDyn Yellowcake Specifications	
				Standard Limit	Reject Limit
U	%	65.0	64.2	≥88.4	≥76.7
Ag	%	< 0.040	< 0.050	0.01	0.04
As	%	< 0.030	0.041	0.01	0.04
Ba	%	< 0.0002	< 0.0002	0.01	0.04
Ca	%	0.12	0.015	0.05	1.00
Cd	%	< 0.001	< 0.001	0.01	0.04
Cr	%	< 0.002	0.0013	0.01	0.04
Fe	%	0.56	0.12	0.15	0.50
K	%	< 0.080	< 0.040	0.20	1.00
Mg	%	0.76	< 0.030	0.02	0.50
Mo	%	0.11	0.32	0.10	0.30
Na	%	0.016	0.019	0.50	3.00
P	%	0.063	< 0.020	0.033	1.31
Pb	%	< 0.020	< 0.020	0.01	0.04
Se	%	< 0.003	< 0.003	0.01	0.04
Ti	%	< 0.030	< 0.020	0.01	0.05
V	%	0.028	< 0.003	0.10	0.75

The uranium concentration is lower than the specification because the uranium precipitates produced were not dried or calcined. Other than uranium, certain metals such as arsenic, iron, magnesium molybdenum, phosphorus and vanadium from one flowsheet or another were higher than their standard limits, implying that some (standard) modifications to the flowsheet, which could not be tested under batch test conditions, will be required for impurity control. These results can only be considered as preliminary as they were done under batch test conditions to provide an indication of solution chemistry.

The above results show that solvent extraction can be used for the uranium recovery unit operation. More definitive testwork for design purposes would involve semi-continuous testwork. As noted elsewhere in this report, the alternative to solvent extraction for uranium recovery is resin-in-pulp. This process, which has specific technical benefits for the Roughrider mineralization, will also require semi-continuous testwork to provide process design criteria.

12.2.7 Leach Residue Settling Tests

Dynamic settling tests were conducted on the Phase I leach residue by Outotec (Canada) Ltd. at SGS Lakefield in May 2009. These tests yielded the following results:

- A feed density of 3.5% solids (w/w) along with the addition of a non-ionic flocculant, MF-351, at a dosage of 438 g/t with the addition of 84 g/t MF-368 coagulant resulted in good settling characteristics and an overflow containing 241 to 174 ppm TSS (Total Suspended Solids).
- Underflow density results for the two tests performed were 29.8% solids (w/w) and 34.8% solids (w/w), though the bed was not able to reach normal operating height due to sample restrictions. These are considered low densities relative to those typical of a counter current decantation ("CCD") circuit.
- Unit thickening areas achieved were 0.17 t/h/m^2 to 0.27 t/h/m^2.
- Rheology values for high rate thickening were low yielding a maximum value of 76 Pa at 34.8% solids (w/w) underflow density.

12.2.8 Tailings Preparation and Effluent Treatment

Tailings and treated effluent were prepared using two flowsheets typically used at northern Saskatchewan uranium operations for tailings treatment.

Tailings solids elemental assays are summarized below in Table 12.7 showing that the elemental analyses for the tailings solids produced by each of the two flowsheets are quite similar.

Table 12.7: Tailings Solids Elemental Analyses

Element	Unit	Flowsheet No. 1	Flowsheet No. 2
As	%	0.0072	0.0071
Ca	%	0.50	1.00
Cu	%	0.019	0.019
Mo	%	0.063	0.054
Ni	%	0.0048	0.0049
Pb	%	0.034	0.035
Se	%	0.0013	0.0012
U	%	0.0074	0.0110
V	%	0.27	0.27
Zn	%	0.0135	0.0165
ABA - NP/AP(1)	-	5.47	11.9

Note: 1. Acid Base Accounting Measurement (Neutralization Potential/Acid Generating Potential)

Key analyses for the treated effluent from each flowsheet tested are summarized in Table 12.8 below. With the exception of molybdenum, all the treated effluent analyzes are low. The molybdenum assays are very high because ferric sulphate additions in each of the flow sheets tested were insufficient to precipitate molybdenum, which had an unexpectedly high concentration in the pregnant leach solution (the pregnant solution assay was not available prior to the treatment test being done). Treatment conditions were adjusted in the second phase of work to match the ferric sulphate additions and pH values which would be typically used during plant operations for this level of concentration in the effluent treatment feed.

Table 12.8: Treated Effluent Analyses

Analyte	Unit	Treated Effluent Flowsheet 1	Treated Effluent Flowsheet 2
As	mg/L	0.04	0.3
Cu	mg/L	0.007	0.1
Mo	mg/L	30.3	9.9
Ni	mg/L	0.02	0.02
Pb	mg/L	0.0001	0.003
Se	mg/L	0.064	< 0.05
U	mg/L	0.013	0.01
Zn	mg/L	0.014	0.21
Ra^{226}(1)	Bq/L	<0.2	<0.5

Note: 1. Radium detection limit higher than normal due to limited sample size.

12.3 Metallurgical Testwork-Phase II

12.3.1 Summary

A second phase of metallurgical testwork has been completed at SGS Lakefield in late 2009 and in 2010 under the direction of Melis using samples from purposed-drilled HQ core (Drillhole No. MWNE-09-085). This second phase of work included composite preparation, comminution (grinding) testwork, confirmation leach tests, solvent extraction tests, uranium precipitation tests, tailings and effluent preparation, and leach residue and tailings settling testwork.

Four variability composites, with uranium grades ranging from 0.11 % U_3O_8 to 16.5% U_3O_8, and one overall composite grading 3.30% U_3O_8 were prepared from the purposed-drilled half core recovered from Hole DDH MWNE-09-85.

Similar to the Phase I test composites, the concentration of impurity elements is relatively low and can be accommodated by appropriate uranium recovery and effluent treatment conditions typically used at northern Saskatchewan uranium mines.

SAG mill power indices and ball mill Bond Work Indices were determined for each of the Phase II variability composites. The SAG mill power indices averaged 14.3 minutes (range of 7.9 to 23.4), the ball mill Bond Work Indices averaged 9.3 kWh/t (range of 7.2 to 11.1), both indicating that the mineralization is softer than average.

A series of leach tests were completed under atmospheric and low pressure leach conditions. Of the ten leach atmospheric pressure leach tests conducted on the overall composite, the highest uranium extraction obtained was 99.2% and the average was 98.5%.

Of the tests on the variability composites the highest extraction was 99.8% for the atmospheric leach of the high grade composite and the lowest extraction was 95.6% for the atmospheric leach of the lower grade composite. The bulk leach test, completed on 35 kg of material assaying 4.16% U_3O_8, yielded a 99.3% extraction for a 12 hour leach. Typical leach conditions are a mesh-of-grind K_{80} of 100 to 125 µm, a leach temperature of 50 °C, a leach retention time of 12 hours or less, a free acid of 15-20 g H_2SO_4/L achieved with an acid consumption of approximately 125 kg H_2SO_4/tonne, and an oxidation reduction potential of 500 mV or greater achieved with an average oxidant consumption of approximately 4 kg $NaClO_3$/tonne.

There was little difference between the solvent extraction isotherms obtained for the strong acid strip process, as used at the Rabbit Lake mill, and the ammonium sulphate strip process as used at the McClean Lake mill. Excellent efficiencies were obtained in both stripping processes. The quality of the uranium precipitate produced from the strong acid strip process met all standards set by the ConverDyn uranium refinery of Englewood Colorado.

Tailings and treated effluent were prepared using two flowsheets typically used at northern Saskatchewan uranium operations for tailings treatment, the same flowsheets used in the Phase I test program. The concentrations of all elements in the treated effluent are below typical regulatory limits set by the provincial and federal governments for Saskatchewan uranium mines. However, the regulations are moving away from assay limits to limits based on environmental loading to the local watershed, which combines assays and volume discharged. Applicable discharge limits for the Roughrider Project will be identified during the formal environmental approval process.

Dynamic settling tests on Roughrider leach residue from the bulk leach test yielded underflow densities of 35% solids (w/w) and a measured thickener unit area of 0.2 t/h/m^2. For the prepared tailings, dynamic settling tests achieved tailings densities of roughly 35% solids (w/w) with thickener rise rates of 5.3 to 9.5 m/h.

12.3.2 Phase II Composite Preparation and Analyses

Four variability composites, Composite Pelitic Gneiss ("PG"), Composite Disseminated Uranium ("DM"), Composite Worm Rock Uranium ("WRM") and Composite Pegmatite ("PE"), as well as an overall composite (Composite RR2) were prepared from the purposed-drilled half core recovered from DDH MWNE-09-85 located in the West Zone of the Roughrider deposit.

The more significant assays for the four variability composites and the overall composite are summarized in Table 12.9 below.

Table 12.9: Phase II Composite Assays

Analyte	Unit	Comp RR2	Comp PG	Comp DM	Comp WRM	Comp PE
U_3O_8	%	3.30	0.19	0.81	16.5	0.11
As	%	0.035	0.0075	0.017	0.080	0.035
Mo	%	0.062	0.022	0.090	0.120	0.015
Co	%	0.0059	< 0.004	< 0.004	< 0.012	0.011
Ni	%	0.022	0.0081	0.013	0.052	0.021
Pb	%	0.16	0.013	0.041	0.79	0.0071
Se	%	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004
V	%	0.13	0.089	0.20	0.19	0.067

There was considerable variation in uranium grade in the variability composites, from a low of 0.11% U_3O_8 in Composite PE to a high of 16.5% U_3O_8 in Composite WRM. The average uranium grade in Composite RR2, an overall composite prepared from DDH MWNE-09-85, was 3.30% U_3O_8, which matched the anticipated overall resource grade for the deposit. Similar to the Phase I test composites the concentration of impurity elements is relatively low and can be accommodated by appropriate uranium recovery and effluent treatment conditions typically used at northern Saskatchewan uranium mines.

12.3.3 Grinding Tests

SAG mill power index and Bond ball mill work indices determined for each of the Phase II variability composites are listed below in Table 12.10. These indices indicate that the Roughrider mineralization is soft compared to other northern Saskatchewan uranium mineralization.

Table 12.10: Summary of Comminution Test Results on Phase II Variability Composites

Composite	SAG Mill Power Index, Min	Bond Ball Mill Work Index, kWh/t
Composite PG	12.8	10.3
Composite DM	7.9	7.2
Composite WRM	12.9	8.7
Composite PE	23.4	11.1
Average	14.3	9.3
Std. Dev.	6.5	1.7

12.3.4 Phase II Leaching Tests

A series of leach tests were completed under atmospheric and low pressure leach conditions, essentially under the same leach conditions used in the Phase I testwork.

Ten leach tests were conducted at atmospheric pressure on the overall Composite RR2 to define leach conditions. Optimum test conditions identified from the atmospheric pressure leach tests were repeated in variability leach tests conducted on the sub-composites PE, DM, PG and WRM. The optimum leach conditions were then repeated on the overall composite RR2 and the sub-composites under low pressure (100 kPa) leach conditions. The low pressure leach tests used oxygen as an oxidant rather than sodium chlorate.

A final bulk leach test was conducted on the remaining mineralization to generate pregnant leach solution for solvent extraction tests, and leached solids for settling tests and tailings preparation.

Leach test results are summarized in Table 12.11 below.

Table 12.11: Summary of Phase II Leach Test Results

Test No.	Composite	% U_3O_8			Final Pregnant Solution			U_3O_8 Extraction (%)
		Direct Head	Calc. Head	Leach Residue	g U_3O_8/L	g Fe^{3+}/L	g Fe^{2+}/L	
AL-1	RR2	3.30	4.12	0.031	17.7	3.60	2.90	99.1
AL-2	RR2	3.30	3.93	0.025	16.5	3.69	3.01	99.4
AL-3	RR2	3.30	3.74	0.047	15.3	1.91	4.69	98.7
AL-4	RR2	3.30	3.26	0.066	14.2	0.79	1.71	98.0
AL-5	RR2	3.30	4.10	0.112	16.5	1.38	4.82	97.3
AL-6	RR2	3.30	3.87	0.027	19.1	2.47	5.06	98.3
AL-7	RR2	3.30	3.92	0.020	18.2	6.94	<0.005	99.2
AL-8	RR2	3.30	4.14	0.018	19.8	7.01	<0.005	98.9
AL-9	RR2	3.30	4.06	0.060	20.0	7.78	0.016	97.5
AL-10	RR2	3.30	4.11	0.040	18.9	4.14	2.06	99.0
VAL-1	PE	0.11	0.11	0.005	0.57	0.43	<0.005	95.8
VAL-2	DM	0.81	0.94	0.021	4.01	1.09	1.31	97.7
VAL-3	PG	0.19	0.13	0.006	0.50	2.50	9.50	95.6
VAL-4	WRM	16.5	15.4	0.024	89.6	1.13	2.27	99.8
LP-1	RR2	3.30	3.78	0.029	16.5	2.98	3.52	99.2
LP-2	PE	0.11	0.13	0.006	0.63	0.19	0.12	95.6
LP-3	DM	0.81	1.12	0.020	5.31	1.51	1.69	98.2
LP-4	PG	0.19	0.16	0.006	0.65	9.64	7.36	96.4
LP-5	WRM	16.5	23.3	0.130	71.9	1.32	1.58	99.4
Bulk Leach	Remaining Feed	4.16	4.10	0.029	20.0	2.99	4.71	99.3

Of the ten leach atmospheric pressure leach tests conducted on Composite RR2, the highest uranium extraction obtained was 99.4% and the average was 98.5%. Of the tests on the variability composites the highest extraction was 99.8% for the atmospheric leach of the high grade Composite WRM (99.4% for the low pressure leach). The lowest extraction was 95.6% for the atmospheric leach of the lower grade Composite PG. Broadly, the feed grades and extractions for the variability composites were correlated, but not linear.

The bulk leach test, completed on 35 kg of material assaying 4.16% U_3O_8, yielded a 99.3% extraction for a 12 hour leach using 19 g H_2SO_4/L free acid and an ORP of 500 mV with 3.9 kg $NaClO_3$/t sodium chlorate addition.

From the leach results it was concluded that leach extractions of 99% or better can be obtained on Composite RR2, with a head grade of 3.30 % U_3O_8. All results show that the Roughrider mineralization is easily leached, confirming leach extractions obtained in the first phase of work. Typical leach conditions are a mesh-of-grind K_{80} of 100 to 125 µm, a leach temperature of 50 °C, a leach retention time of 12 hours or less (some tests reached maximum uranium extraction after only six hours of leaching), a free acid of 15-20 g H_2SO_4/L achieved with an acid consumption of approximately 125 kg H_2SO_4/tonne, and an oxidation reduction potential of 500 mV or greater achieved with an average oxidant consumption of approximately 4 kg $NaClO_3$/tonne.

Analysis of the pregnant leach solution from the bulk leach is listed below in Table 12.12.

Table 12.12: Phase II Bulk Leach Test Pregnant Leach Solution Assays

Analyte	Unit	Assay
Mg	g/L	0.62
Mn	g/L	0.1
Na	g/L	0.42
Ni	g/L	0.088
P	g/L	0.04
Pb	g/L	<0.02
Sb	g/L	<0.003
Se	g/L	n/a
Sn	g/L	<0.003
Sr	g/L	0.003
Ti	g/L	<0.006
Tl	g/L	<0.003
V	g/L	0.076
W	g/L	<0.003
Y	g/L	0.051
Zn	g/L	<0.002

On a percent U basis (% / % U) the molybdenum content of this pregnant leach solution (0.008%) is one order of magnitude lower than the Phase I bulk leach solution (0.07%) which was generated from a lower uranium grade composite (0.64% U_3O_8) containing 0.17% Mo.

12.3.5 Phase II Solvent Extraction Tests

Solvent extraction ("SX") consists of an extraction process and a stripping process.

The extraction process consists of contacting and vigorously agitating the pregnant leach solution, prepared in the bulk leach test and diluted somewhat (to 11.2 g U_3O_8/L) in the dynamic settling tests, with a kerosene-based organic containing an amine and isodecal alcohol. The amine, Alamine 336 in these tests, absorbs the uranium. The kerosene, Isopar-M in these tests, act as a diluent for the amine. The isodecal alcohol, more familiarly known as Isodecanol, speeds up the separation of the pregnant leach solution/organic emulsion formed during the agitation.

Two stripping processes were tested during this procedure: one using a strong sulphuric acid (400 g H_2SO_4/L) strip and one using an ammonium sulphate [180 g $(NH_4)_2SO_4$/L] strip at pH 4.5 controlled with ammonia. The loaded strip solutions produced by each process are chemically different and the uranium contained in these solutions must be precipitated using different chemical processes. In addition, the different composition of the loaded strip solutions required different tailings and by-product unit operations.

Figure 12.3 below plots the extraction isotherms for the strong acid and ammonium sulphate stripping processes. From this graph it can be seen that there is little difference between the extraction isotherms obtained for the strong acid strip process and the ammonium sulphate strip process.

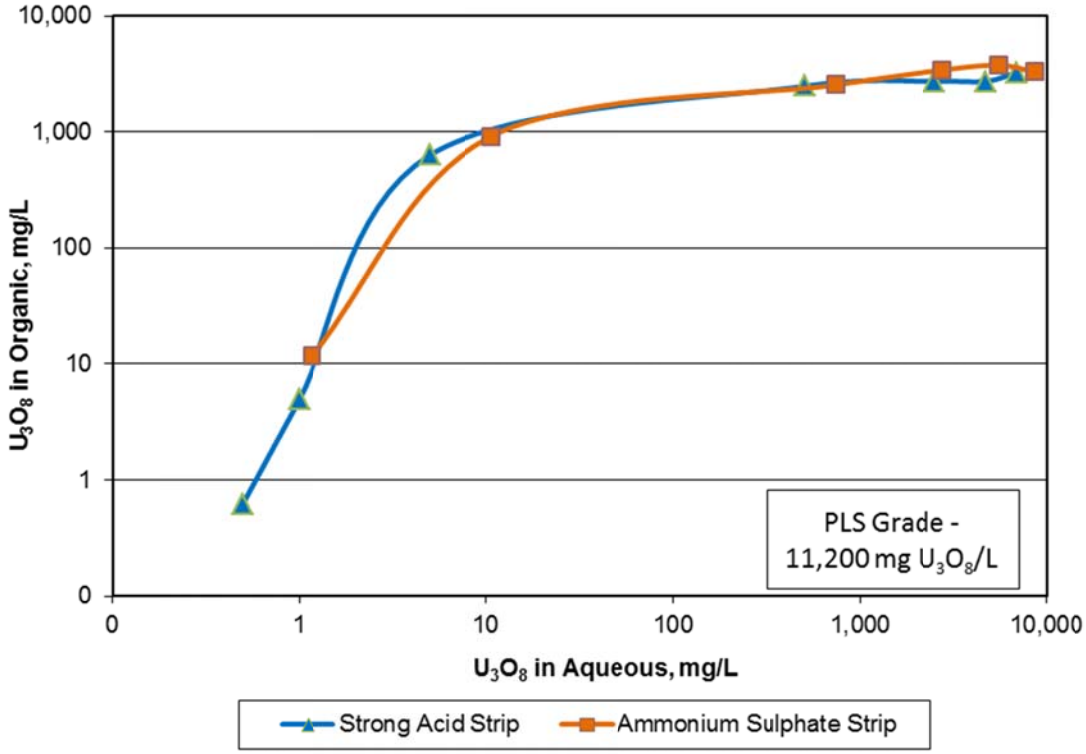


Figure 12.3: Strong Acid and Ammonium Sulphate Extraction Isotherms

12.3.6 Uranium Precipitation

The pregnant strip solution from the strong acid strip and the ammonium sulphate strip was submitted to uranium (yellowcake) precipitation.

In the strong acid strip process pregnant strip liquor is first treated with lime to remove excess sulphate ions as calcium sulphate (gypsum), and then the uranium is precipitated with hydrogen peroxide using magnesium oxide for pH control. For the ammonium sulphate strip process pregnant strip liquor is precipitated with ammonium at pH 7.

In the process used for the strong acid strip solution, hydrogen peroxide precipitation, the efficiency of uranium precipitation was 99.99%. In the process used for the ammonium sulphate strip solution, precipitation with ammonia, the efficiency of uranium precipitation was 99.995%, essentially 100% to the limit of the detection efficiency for uranium.

The yellowcake products were analyzed with available analyses summarized in Table 12.13 below.

Table 12.13: ConverDyn Uranium Refinery – Englewood Colorado November 2010 Example of Uranium Concentrate Specifications vs. Roughrider Phase II Yellowcake Analyses

Component	"Standard Concentrates"	"Maximum Limit Concentrates"	Strong Acid Strip Precipitate	Ammonium Sulphate Strip Precipitate
Uranium (U)	75%	65% min.	69.2[1]	64.8[1]
Arsenic (As)	0.01%	0.04%	-	<0.004
Barium (Ba)	0.01%	0.04%	-	<0.0002
Cadmium (Cd)	0.01%	0.04%	-	<0.001
Calcium (Ca)	0.05%	1.00%	0.031	0.0067
Chromium (Cr)	0.01%	0.04%	<0.001	<0.0004
Iron (Fe)	0.15%	0.50%	0.12	0.17
Lead (Pb)	0.01%	0.04%	<0.02	-
Magnesium (Mg)	0.02%	0.50%	<0.10	<0.02
Molybdenum (Mo)	0.10%	0.30%	0.0019	0.21
Potassium (K)	0.20%	1.00%	<.0.05	<0.04
Selenium (Se)	0.01%	0.04%	<0.003	-
Silver (Ag)	0.01%	0.04%	<0.05	-
Sodium (Na)	0.50%	3.00%	0.016	0.012
Titanium	0.01%	0.05%	<0.03	<0.02
Vanadium (V)	0.10%	0.75%	<0.04	<0.05

Note: 1. Batch test result only, higher uranium content typically achievable

The quality of the uranium produced by the strong acid strip process met all standards set by the ConverDyn uranium refinery of Englewood Colorado. Drying of the strong acid strip precipitate would also increase the uranium concentration due to removal of some of the water of hydration.

12.3.7 Tailings Preparation and Effluent Treatment

Tailings and treated effluent were prepared using two flowsheets typically used at northern Saskatchewan uranium operations for tailings treatment, the same flowsheets used in the Phase I test program.

Tailings solids elemental assays are summarized below in Table 12.14 showing that the elemental analyses for the tailings solids produced by each of the two flowsheets are quite similar, as observed in the Phase I testwork.

Table 12.14: Phase II Tailings Solids Elemental Analyses

Element	Unit	Flowsheet No. 1	Flowsheet No. 2
As	%	0.028	0.026
Ca	%	0.74	1.30
Cu	%	0.0073	0.0072
Mo	%	0.058	0.054
Ni	%	0.0075	0.0076
Pb	%	0.22	0.20
Se	%	0.0019	0.0019
U_3O_8	%	0.021	0.020
V	%	0.14	0.14
Zn	%	0.002	0.002

Key analyses for the treated effluent from each flowsheet tested are summarized in Table 12.15 below. Radium analyses were not available for these tests, but the Phase I testwork showed that acceptable radium levels in treated effluent can be reached.

Table 12.15: Phase II Treated Effluent Analyses

Analyte	Unit	Treated Effluent Flowsheet 1	Treated Effluent Flowsheet 2
As	mg/L	<0.3	<0.3
Cu	mg/L	<0.04	<0.04
Mo	mg/L	0.0412	0.03
Ni	mg/L	<0.06	0.06
Pb	mg/L	<0.01	<0.01
Se	mg/L	<0.3	<0.3
U_3O_8	mg/L	0.035	0.012
Zn	mg/L	<0.1	<0.05

The concentrations of all elements are below typical regulatory limits set by the provincial and federal governments for Saskatchewan uranium mines. However, the regulations are moving away from assay limits to limits based on environmental loading to the local watershed, which combines assays and volume discharged. Applicable discharge limits for the Roughrider Project will be identified during the formal environmental approval process.

12.3.8 Settling Tests on Leach Residue and Tailings

Outotec (Canada) Ltd. was contracted to conduct dynamic settling tests on Roughrider leach residue from the bulk leach test and on simulated tailings.

With dilute feed [2.5% solids (w/w)] and flocculant (Magnafloc 351) additions of 75 g/t, underflow densities of 35% solids (w/w) were achievable for the leach residue yielding a thickener unit area of 0.2 $t/h/m^2$. The rheology for high rate thickening was a maximum of 86 Pa at a 37.3% solids (w/w) underflow density, indicating that a normal rake and rake motor would be sufficient for thickener design. The leach residue results achieved were similar to those achieved in the Phase I testwork.

For the prepared tailings a dilute feed and up to 150 g/t flocculant (Magnafloc 338) addition achieved tailings densities ranging from 26% solids (w/w) to 38% solids (w/w) with thickener rise rates of 5.3 to 9.5 m/h. The rheology for the compacted slurry had a maximum value of 88 Pa at a 36.9% solids (w/w) underflow density, again indicating that a normal rake and rake motor would be sufficient for thickener design.

Paste simulation tests performed on the tailings samples produced underflow densities of 40 % solids (w/w) to 41 % solids (w/w) with unsheared vane yield stresses of 72 Pa to 134 Pa, indicating that the tailings material could be compressed further.

12.4 Work In Progress

A third phase of variability testwork is being completed at SGS Lakefield on core samples from purpose-drilled holes (DDH Nos. 09-171 and 09-172) from the Roughrider West Zone under the direction of Melis. Eight variability composites and one overall composite have been prepared, comminution tests performed, and scoping leach tests carried out. The obtained results, which will be reported as part of the project development along with additional tailings and treated effluent data, are similar to those achieved in the Phase II testwork.

A fourth phase of testwork has been initiated under the direction of Melis with the receipt of additional test samples from a purpose-drilled hole (DDH MWNE-11-718) in the Far East Zone of the Roughrider deposit. Four variability composites and one overall composite will be prepared for testing. This test program will include confirmation and variability comminution tests, confirmation and variability leach tests, uranium recovery tests using resin-in-pulp (an alternative process to solvent extraction having specific technical and economic advantages for the Roughrider deposit), yellowcake precipitation of the liquor from elution of the loaded resin, and further tailings preparation and environmental analyses.

As the project develops the final phase of testwork will include semi-continuous testing of the selected process.

13 Mineral Resource Estimate

13.1 Introduction

Hathor commissioned SRK to prepare an updated mineral resource estimate for the Roughrider West Zone on September 7, 2010. This represents the second mineral resource evaluation prepared for this deposit. A previous mineral resource statement was prepared in 2009 by Scott Wilson RPA Inc., as documented in a technical report dated December 18, 2009.

The resource estimation work was completed by G. David Keller, P. Geo (APGO#1235) with the assistance of Dominic Chartier, P. Geo (APQ#874), both "independent qualified persons" for the purpose of National Instrument 43-101.

Subsequently, the resource estimation work for the Roughrider East Zone was completed by G. David Keller, P.Geo. (APGO#1235) and Sébastien Bernier, P.Geo. (APGO#1847), both appropriate "independent qualified persons" as this term is defined in National Instrument 43-101. The effective date of the resource statement is May 17, 2011.

13.2 Resource Database

13.2.1 Roughrider West Zone

Drillhole data used to evaluate the mineral resources for the Roughrider West Zone was provided as MS Excel spreadsheet exports from the Hathor drillhole database containing all information for 123 diamond drillholes comprising approximately 3,950 sample intervals assayed for U_3O_8 and other metals including cobalt, copper molybdenum, nickel and selenium. Each drillhole interval was coded by lithology and alteration intensity. The database also included 264 specific gravity measurements.

The drillhole data provided by Hathor was validated by:

- Reviewing collar and down hole survey data;
- Checking the minimum and maximum values for each field in the drillhole database and confirming those values outside of expected values;
- Checking for gaps, overlaps and out of sequence intervals; and
- Generating drillholes in Datamine and then reviewing drillholes on a section by section base to ensure that mineralization and alteration are consistent with drilling.

Six boreholes were removed from the database because of down hole survey problems. The holes removed from the database are:

- MWNE-09-132;
- MWNE-09-133;
- MWNE-09-134;
- MWNE-09-135;
- MWNE-09-137; and
- MWNE-09-142.

Three drillholes used for metallurgical testing were used for modelling the boundaries of uranium mineralization and defining resource domains utilizing downhole gamma data as a proxy for uranium grade:

- MWNE-09-171;
- MWNE-09-172;
- MWNE-09-085.

These drillholes were not used to estimate metal grades.

After reviewing the electronic data from the West Zone area, SRK is of the opinion that drillhole database for the project is sufficiently reliable to interpret with confidence the boundaries of the uranium mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

13.2.2 Roughrider East Zone

Drillhole data used to evaluate the mineral resources for the Roughrider East Zone were provided as CSV exports from the Hathor drillhole database containing all information for twenty-one diamond drillholes comprising approximately 368 sample intervals assayed for U_3O_8 and other metals including cobalt, copper, molybdenum, nickel and selenium. Each drillhole interval was coded by lithology and alteration intensity. The database also included 46 specific gravity measurements.

The drillhole data provided by Hathor was validated by using the same methods used for the West Zone:

After reviewing the digital data for the Roughrider East Zone, SRK is of the opinion that the drillhole database for the East Zone is sufficiently reliable to interpret with confidence the boundaries of the uranium mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

13.3 Interpretation and Modelling

13.3.1 Roughrider West Zone

The Roughrider West Zone is typified by a central high grade core of mineralization within a lower grade domain of mineralization. The entire low grade and high grade domains are contained within in a large envelope of alteration that is roughly bounded by the HHW, FWW and MWD units with the upper extent of moderate to intense alteration bounded approximately by the Athabasca Sandstone unconformity.

Hathor provided SRK with an initial sectional interpretation of the high and low grade domains as well as an alteration envelope for the Roughrider West Zone. This interpretation was based on uranium assays, alteration intensity and a sub-horizontal structural trend that influences mineralization. Section lines were generally orientated northwest-southeast.

To assist modelling the high grade and low grade domains, SRK generated wireframe grade shells based on U_3O_8 assays and alteration intensity using LeapFrog software. A series of grades and alteration intensity shells were generated in LeapFrog to model shells for a high grade core and low grade domains for uranium assays and alteration values. LeapFrog shells were modelled unconstrained and constrained by the HWW, FWW and MWD units. The following shells were generated with LeapFrog:

- High grade domain, 1.0, 3.0 and 5.0 percent U_3O_8;
- Low grade domain, 0.01, 0.05, and 0.1 percent U_3O_8; and
- Alteration, 3, 3.5, 4, 5 and intensity values.

LeapFrog modelling indicated that the best threshold model for a high grade domain is the 3.0% U3O8. Two LeapFrog shells with thresholds of 0.01 and 0.05% U3O8 were used to help define the low grade domain. An alteration intensity threshold of 4.0 was provided as a secondary inference for the high grade domain as well. The unconstrained and constrained models did not significantly change the grade shell trends.

SRK modelled a high grade and an enclosing low grade domains using GEMS software. Interpreted polylines are orientated approximately northwest-southeast and interpreted at a sectional spacing of about 5 m to 10 m. Sectional interpretations were extended from 20 m to 25 m at the peripheries of interpreted sub-zones.

For the high grade domain a 3.0% threshold was used to group high grade assays together into geological sub-zones, this grouping includes intervening lower grade intervals between or next to high grade intersections. Similarly, low grade domain was modelled using a 0.10% to 0.05% threshold value. Polylines were generated by group areas with more or less consistent values above threshold values and excluding areas with spotty low grades. In some cases lower grade material was included in the interpreted polylines. The majority of polyline outlines were snapped to drillhole intersections. In a few cases polylines were interpreted between drillhole intersections to ensure a geologically correct three dimensional interpretation.

GEMS polylines were imported into Gocad software to generate wireframe solids. SRK generated 11 sub-zones within the high domain sub-zone and four sub-zones within the low grade domain for the Roughrider West Zone. High and low grade zones were assigned numerical codes as outlined in Table 13.1. All high grade sub-zones are contained in one main low grade wireframe (Zone 100) solid Figure 13.1. All 15 sub-zones were considered separately in statistical and variogram analysis.

Table 13.1: Resource Domains and Zones for the Roughrider Uranium Deposit

Domain	Sub-Zone
High Grade	1, 2, 3, 4, 5, 6, 7, 8, 9, 10, and 11
Low Grade	100, 200, 300, and 400

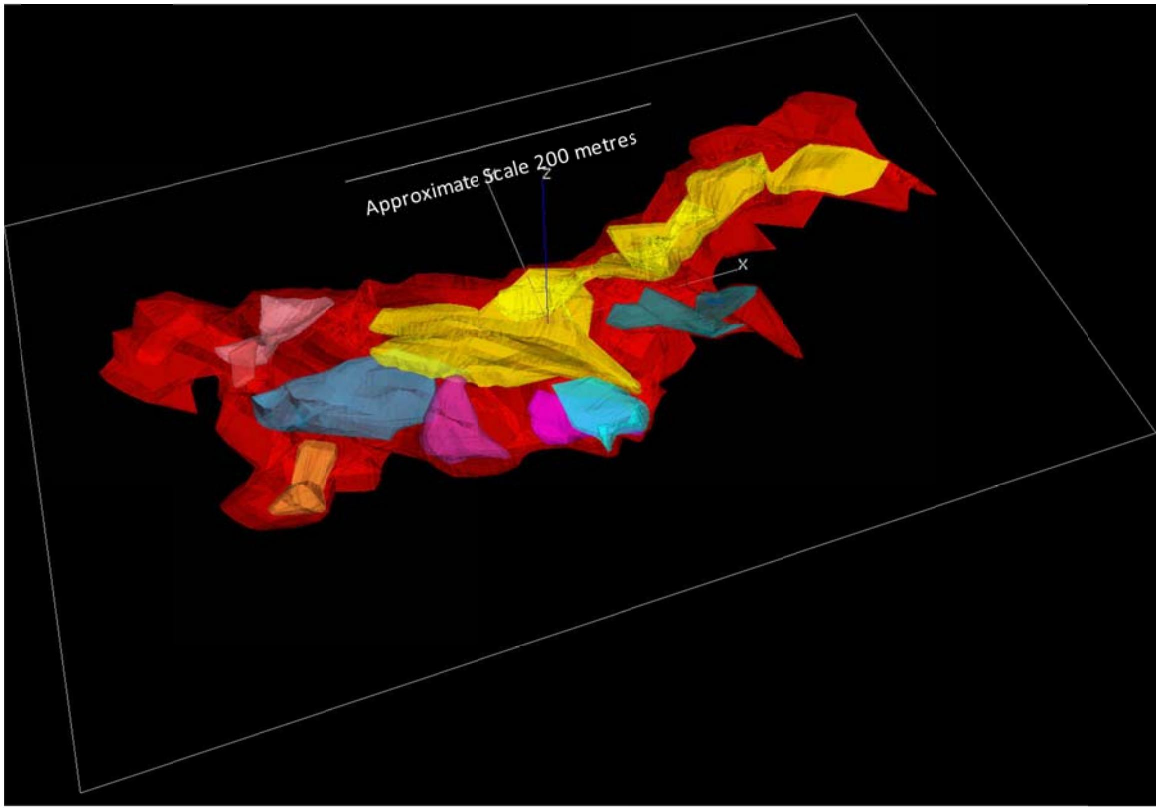


Figure 13.1: Roughrider West Zone Low Grade Zone 100 and High Grade Sub-Zones. View Looking Northeast

13.3.2 Roughrider East Zone

The boundaries for uranium mineralization were initially modelled as a wireframe solid by Hathor using 15 m to 20 m spaced sections and a cut-off grade of 0.5% U_3O_8. The Hathor interpretation was is based on a structural interpretation of the Midwest Trend and Roughrider Corridor structures. Hathor outlined seven high grade U_3O_8 sub-zones. Hathor used MapInfo-Discover to generate the wireframe solid interpretation.

SRK modified the Hathor interpretation using GoCAD mining software. SRK also generated LeapFrog grade shells for the sub-zone using a threshold grade of 0.5% U_3O_8 to assist in modelling the deposit.

SRK made minor modifications to the original model based on the Leapfrog shells, snapping wireframe contacts to drillhole sample intervals and some minor changes in structural orientations of a few sub-zones.

Seven wireframe solids modified by SRK were used for resource evaluation of the Roughrider East Zone. The final interpreted wireframe solids for the Roughrider East Zone are outlined in Figure 13.2.

Low grade mineralization below 0.5 percent U_3O_8 is present in the Roughrider East Zone; however, it appears to be patchy and discontinuous and was therefore not modelled by Hathor or SRK.

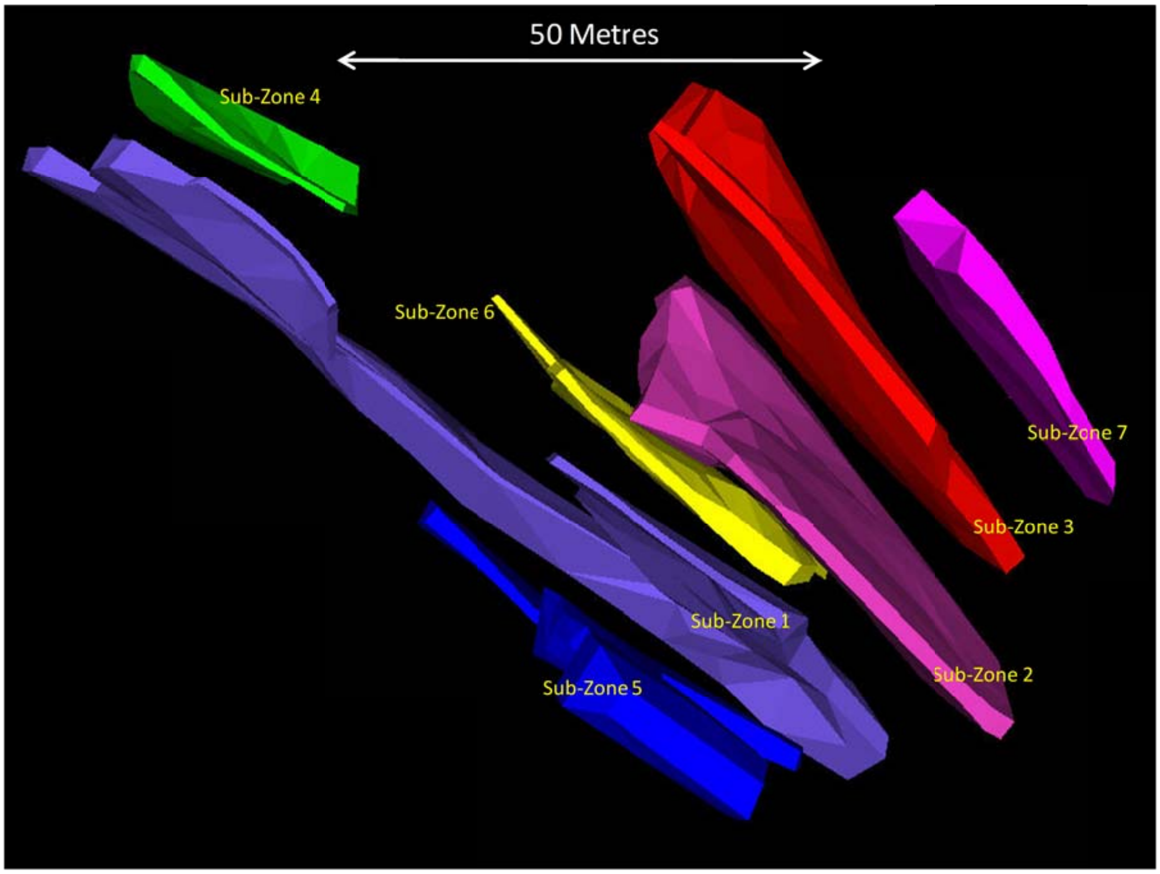


Figure 13.2: Three dimensional View of the Roughrider East High Grade Sub-Zones (Looking Northwest)

13.4 Specific Gravity

13.4.1 Roughrider West Zone

Specific gravity is significantly variable in this deposit. The main factors likely influencing the specific gravity are the degree of alteration and the amount of uranium mineralization.

Increasing alteration is associated with lower specific gravity as the rock minerals are altered to clay minerals. Increasing amounts of uranium mineralization increases specific gravity as more of the massive metal is present. A scatter plot of combined domains, uranium assays and density measurement pairs shows a flat trend for low U_3O_8 grades.

The slope of the relationship increases sharply above grades of about 3.0% indicating a very significant change in the relationship with higher grade uranium mineralization. In both cases there is a significant dispersion of values about each trend indicating a significant variability in the data sets.

A plot of specific gravity and U_3O_8 grades for the high grade domain shows a good correlation between high grade uranium assays and increasing specific gravity values that can be modelled using a polynomial (power of two) regression. The relationship between U_3O_8 grades and specific gravity is quite strong as indicated by correlation and rank correlation factors of 0.89 and 0.85.

It is possible to calculate specific gravity values for each U_3O_8 assay with a paired specific gravity sample for the high grade domain and then use the combined measured and calculated values to estimate specific gravity for the high grade sub-zones. SRK believes this approach is too simplistic as it does not address possible spatial variations of specific gravity and the significant disparity between the assay and specific gravity data sets, 922 to 152 respectively for the high grade domain.

SRK concludes that specific gravity for the deposit must be estimated as the data set is too variable to dataset averages. Further despite a strong correlation of U_3O_8 grades and specific gravity it is not appropriate to use a simple regression relationship to back calculate specific gravity from assay data.

13.4.2 Roughrider East Zone

Specific gravity varies significantly in the Roughrider East Zone similar to the West Zone. Similarly increasing alteration is associated with lower specific gravity as the original minerals are altered to clay minerals. Increasing amounts of uranium mineralization increase specific gravity as more of the massive metal is present. East Zone data follows a similar trend with a small flat trend for low U_3O_8 grades and a sharp upward trend above grades of about 3.0. There is a significant dispersion of values about this trend.

There is a good correlation between specific gravity and U_3O_8 composites for the high grade sub-zone. Although a strong relationship exists between U_3O_8 grades and specific gravity the disparity between the sizes of data sets is significant, 401 to 46 composites, respectively. This precludes calculating specific gravity from the relationship observed with uranium oxide grades.

13.5 Core Recovery

The average core recovery for the Roughrider West Zone is 98% and 93% for the East Zone. SRK examined core loss intervals for both sub-zones. This examination although qualitative indicated that U_3O_8 assays from sub-zones with significant core loss correlated reasonably with gamma peaks and that higher assay values are not being smeared into low grade areas. Based on this and the generally high core recovery in mineralized sub-zones for both East and West Zones U_3O_8 assay values for sub-zones with poor recovery do not represent a significant bias and therefore can be used directly for the estimation of mineral resources.

13.6 Compositing

13.6.1 Roughrider West Zone

Assay and specific gravity data must have common support for statistical analysis, variography and estimation. A composite length of 0.5 m was chosen for both domains given 94% to 96% of sample intervals are less than or equal to 0.5 m. To provide this common support, all intervals within the 15 sub-zones were composited separately using this composite length.

As a majority of samples were collected at 0.5 m compositing has a limited effect.

13.6.2 Roughrider East Zone

Similarly, assay and specific gravity data must have common support for statistical analysis, variography and estimation. The compositing length for the Roughrider East Zone is based on sample length histograms for the combined seven high grade sub-zones. A composite length of 0.5 m was chosen as 91% of sample intervals are less than or equal to 0.5 m. To provide this common support, all intervals within the seven sub-zones were composited separately using this composite length.

13.7 Capping

13.7.1 Roughrider West Zone

SRK evaluated impact of high grade outliers for the U_3O_8 composite database for both high grade and low grade domains. Of the 15 sub-zones only eight sub-zones were considered for capping using cumulative probability plots, histograms and capping sensitivity plots. However after examining the spatial distribution of higher grades with respect to other drillholes and adjacent composites for the eight sub-zones, SRK concludes that no significant outliers are present in the database because high grades above the 95 to 98 percentiles for each of the sub-zones are supported by adjacent composites or composites in nearby drillholes that show a progression of higher grades within ten to 20% of the U_3O_8 capping value.

A review of specific gravity composites for sub-zones 5 and 100 using cumulative probability plots and spatial distribution of high values and indicates that all higher specific gravity values are supported by nearby high values as well as high grade U_3O_8 values. SRK concludes that no significant outliers are present and capping is not required for specific gravity data. All other zones contained less than twenty composites and are not considered for capping.

Potentially deleterious metals (arsenic, cobalt, copper, molybdenum and nickel and selenium) are not capped.

13.7.2 Roughrider East Zone

SRK evaluated the impact of U_3O_8 composite outliers for each sub-zone using cumulative probability plots, histograms and examining the spatial distribution of higher grades with respect to other drillholes and adjacent composites. Outlier analysis was also conducted for deleterious metals.

SRK concludes that no significant outliers are present in the data distributions because high grades above the 95 to 98% for each sub-domain resource domain are supported by adjacent composites or composites in nearby drillholes with grades ranging from 20.0% to 40.0% U_3O_8. Specific gravity data is limited and not very variable and therefore not capped.

Potentially deleterious metals (arsenic, cobalt, copper, molybdenum and nickel and selenium) are not capped.

13.8 Variography

13.8.1 Roughrider West Zone

Traditional, normal scores and normal scores variograms were used to model the spatial distribution of U_3O_8 and specific gravity composites. Variograms were developed for zones 5 and 100 for U_3O_8 analysis as all other sub-zones contained too few composites for analysis or no models were discerned.

As specific gravity data was more limited than U_3O_8 composites, variograms for sub-zones 5 and 100 were poorly defined. SRK developed specific gravity variogram models based on combined sub-zones for either the low or high grade domains. Using a traditional variogram models were developed for low grade domains without being able to discern anisotropy between major and semi major axis. A normal scores variogram was used to develop an omni-directional model for high grade specific gravity composites. Additionally, normal scores variogram was used to model the product of U_3O_8 times specific gravity paired data.

All variograms were found to be orientated along a 45 dip direction to the north or were found to be omni-directional. Possible plunges or rakes to the north dip direction were not well defined.

Variogram models are summarized in Table 13.2.

Variogram analysis was not conducted on potentially deleterious elements.

Table 13.2: Summary of Variogram Models for Roughrider West

Variable	Domain	Sub-Zone	C0	CC	Model	Rx (m)	Ry (m)	Rz (m)	Z Axis	X Axis	Z Axis	Comments
U_3O_8%	Low Grade	100	0.1	0.70	Exponential	15	7	4	0	40	0	Normal Scores
				0.20	Spherical	15	20	15				
	High Grade	5	0.2	0.65	Exponential	5	5	2	0	40	0	
				0.15	Spherical	20	20	2				
Specific Gravity	Low Grade	100-400	0.05	0.65	Exponential	5	5	22	0	40	0	Isotropic in Plane
				0.25	Spherical	20	20	22				
				0.05	Spherical	30	30	22				
	High Grade	01-11	0.1	0.20	Exponential	5	5	5	0	0	0	Omni-directional
				0.70	Spherical	35	35	35				Normal Scores Correlogram

13.8.2 Roughrider East Zone

Traditional and normal scores variograms were used to model the spatial distribution of U_3O_8 composites. A single variogram was developed for the combined sub-zones, as each sub-zone contains too few composites for analysis. Variogram analysis was not conducted on potentially deleterious elements. There is insufficient specific gravity data for variogram analysis. The U_3O_8 variogram is orientated parallel to the general strike and dip-direction of the sub-zones.

The U_3O_8 variogram model was assumed for the estimation of potentially deleterious elements and specific gravity excluding sub-zones 4, 5 and 7. Variogram model parameters are outlined in Table 13.3.

Table 13.3: Summary of Variogram Models

Variable	Domain	Zone	C_0	CC	Model	Rx [m]	Ry [m]	Rz [m]	Datamine Rotation		Comments
									Z Axis	Y Axis	
U_3O_8%	All	All	0.20	0.15	Exponential	15	15	3	130	45	Normal Scores
				0.65	Spherical	30	30	9			

13.9 Block Model Parameters

A sub-blocked for both West and East models was generated using Datamine Studio 3. The block model coordinates are based on the local UTM coordinate grid (NAD83, Zone 13). The parent block size is 4 m by 4 m by 2 m and is sub-blocked to 1.0 m in the X and Y directions and 0.002 m in the Z direction. The definition of the Roughrider West Zone block model is presented in Table 13.4. and Table 13.5 for the East block model.

Criteria used in the selection of block size include the borehole spacing, composite assay length, consideration for the potential size of the smallest mining unit and the geometry of the modelled auriferous sub-zones. A block model was generated for each of the 15 wireframe sub-zones.

Table 13.4: Roughrider West Zone Block Model Definition

Axis	Block Size (m)	Origin (m)	Extent (m)	No. Blocks
X	4	556,000	557,200	75
Y	4	6,466,750	6,467,350	38
Z	2	150	450	75

Table 13.5: Roughrider East Zone Block Model Definition

Axis	Block Size (m)	Origin (m)	Extent (m)	No. Blocks
X	4	556,000	557,200	117
Y	4	6,466,750	6,467,350	35
Z	2	150	450	50

13.10 Estimation

13.10.1 Roughrider West Zone

The estimation strategy for the Roughrider West Zone Deposit consists of estimating U_3O_8 for each of the Roughrider sub-zones separately using only composites from each sub-zone.

U3O8 grades were estimated using three estimation runs. The first estimation run is based on a search ellipse with ranges equal to the largest variogram model structure and the second run consists of a search ellipse range equal to twice the variogram range. The third estimation run (not required for sub-zones 100 and 200) consist of a search ellipse generally three times the search ellipse range. The bulk of blocks are estimated in the first run. Second and third estimation runs adds only about 18% and 9% more material, respectively to ensure that all blocks in the resource domains are estimated. The estimation parameters including estimation runs and search ellipses sizes are summarized in Table 13.6.

Specific gravity presents more of a challenge as for sub-zone 4 and there are no specific gravity composites. As well, sub-zone 300 and 400 contain only one composite.

Ordinary kriging was used to estimate specific gravity for most of the sub-zones. For domains with limited specific gravity data, global kriging values were substituted for unestimated values (sub-zones 2, 6, 400). Global kriging values of the entire high or low grade data sets were used for sub-zones without density data (sub-zone 4, 10, 300).

Potentially deleterious elements arsenic, cobalt, copper, molybdenum, and nickel were estimated using ordinary kriging. Variogram models for U_3O_8 were assumed for these metals. The same estimation parameters used for U_3O_8 were used for estimating these elements.

Table 13.6: Summary of Estimation Parameters for West Zone

Variable	Domain	Zone	Run	Min	Max	Octant Search	Rx (m)	Ry (m)	Rz (m)	Z Axis*	X Axis*	Z Axis*	Max Comp. per hole
U_3O_8%	Low Grade	100-400	1	3	12	Yes	15	20	15	0	40	0	3
			2‡	3	12	Yes	30	40	30				3
			3‡	2	12	No	45	60	45				3
	High Grade	1-11	1	3	12	Yes	20	20	2	0	40	0	3
			2	3	12	Yes	40	40	4				3
			3	3	12	No	60	60	6				3
Specific Gravity	Low Grade	100-400	1	2	12	Yes†	30	30	22	0	40	0	3
			2	2	12	Yes†	60	60	44				3
			3	3	12	No	90	90	66				3
	High Grade	1-11	1	2	12	Yes†	35	35	35	0	0	0	3
			2	2	12	Yes†	70	70	70				3
			3			No	88	88	88				3

* Datamine rotations.
‡ Not required for Zone 100 and 200.
† Number of octants filled two for an estimate, otherwise three.

13.10.2 Roughrider East Zone

The estimation strategy for the Roughrider East Zone consists of estimating U_3O_8, potentially deleterious metals (arsenic, cobalt, copper, molybdenum, nickel and selenium) and specific gravity into a block model for each of the seven sub-zones independently. Specific gravity was not estimated for sub-zones 4, 5 and 7. Sub-zones 4 and 5 have only two and four composites, respectively. There is no specific gravity data for sub-zone 7.

U_3O_8 grades were estimated using three estimation runs using ordinary kriging using composite data from each domain, separately. The first estimation run considers a search ellipse with ranges equal to the largest variogram model structure. The second run considers a search ellipse equal to twice the variogram ranges while for the third estimation run the search ellipse was inflated to four times the variogram ranges. The bulk of blocks are estimated by the first run. The second and third estimation runs allow only about 10% and 12% more material, respectively to ensure that all blocks in the resource domains are estimated. Estimation parameters are summarized in Table 13.7. Estimation of specific gravity using composites provides the most reasonable results maintaining the variability of the original composites.

Specific gravity was estimated using an inverse distance squared function. For sub-zone 4 the average of two specific gravity composites (2.14) was assigned to all blocks of that domain. Blocks from sub-zone 5 were all assigned as specific gravity value of 2.23, the only data available for that domain. The average of all specific gravity composites (2.74) was assigned to all blocks in sub-zone domain 7.

Potentially deleterious elements (arsenic, cobalt, copper, molybdenum, nickel and selenium) were estimated using ordinary kriging. Variogram models for U_3O_8 were assumed for these metals. The same estimation parameters used for U_3O_8 were used for estimating these elements.

Estimates were verified by conducting checks on sub-zone 2. Verification procedures included visual examination of block grades to drillhole composites, and comparing estimated grades at zero cut-off to nearest neighbour estimates and declustered means for each sub-zone. All validation checks confirm that the block estimates are appropriate and reflect the underlying borehole sampling data.

Only parent blocks were estimated. All sub-blocks were assigned parent block values.

Table 13.7: Summary of Estimation Parameters for East Zone

Variable	Estimator	Domain	Run	Min.	Max.	Octant Search	Search Volume			Rotation		Maximum Composite per Hole
							X (m)	Y (m)	Z (m)	Z Axis*	Y Axis*	
U_30_8%	OK	1-7	1	2	12	Yes†	30	30	9	130	45	3
			2	2	12	Yes†	60	60	18			3
			3	3	12	No	120	120	27			3
As, Co, Co, Mo, Ni, Se	OK	1-7	1	2	12	Yes†	30	30	9	130	45	3
			2	2	12	Yes†	60	60	18			3
			3	3	12	No	120	120	27			3
Specific Gravity	ID2	1, 2, 3,6	1	2	12	Yes‡	30	30	9	130	45	3
			2	2	12	Yes‡	60	60	18			3
			3			No	120	120	120			3

* Datamine Rotations
† Minimum 3 octants, minimum 2 composite per octant
‡ Minimum 2 octants, minimum 1 composite per octant

13.11 Mineral Resource Classification

13.11.1 Roughrider West Zone

Block model quantities and grade estimates for the Roughrider West Zone Deposit have been classified according to the “CIM Standards on Mineral Resources and Reserves: Definition and Guidelines” (December, 2005) by G. David Keller, P. Geo (APGO#1235) an “independent qualified person” as defined by National Instrument 43-101.

Classification criteria include drilling density, variography results and estimation run. All blocks in Zone 100 are classified as Indicated because this sub-zone is well informed by drilling spaced at 5.0 m to 10.0 m and blocks were estimated with U_3O_8 grades entirely by the first estimation run. Zone 5 is also well informed by data spaced from 5.0 m to 10.0 m with the exception of the extreme northwest portion of the Zone that is not well informed by drilling. This area was manually assigned an Inferred classification. Zone 2 was classified according to estimation run; blocks classified in the first estimation run was classified as Indicated with all other blocks were classified as Inferred. All other sub-zones were classified as Inferred because they are informed by limited data.

13.11.2 Roughrider East Zone

Mineral resources for the Roughrider East Zone have been classified according to the “CIM Definition Standards for Mineral Resources and Mineral Reserves” (December, 2005) by G. David Keller, P. Geo (APGO#1235) and Sébastien Bernier P.Geo. (APGO#1847) both “independent qualified person” as defined by National Instrument 43-101.

After review, SRK considers that all modelled blocks in the Roughrider East Zone should be classified as Inferred within the meaning of CIM definitions because the confidence in the estimates is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure and justify an Indicated classification. Additional infill drilling and sampling is required to support a higher classification. It cannot be assumed that all or any part of an Inferred mineral resource will be upgraded to an Indicated or Measured mineral resource as a result of continued exploration.

13.12 Model Validation

13.12.1 Roughrider West Zone

The block model prepared by SRK was validated using sub-zone 5 and sub-zone 100 as tests of estimation procedures. U_3O_8 grades block estimates using ordinary kriging were checked by:

- Comparing drillhole composites to block model estimates;
- Comparing declustered means to the average grade block model grade;
- Comparing average grades for nearest neighbour and inverse distance (power of two) estimates at no cut-off grade to average block; and
- Swath plots of nearest neighbour, inverse distance squared ordinary kriging.

Ordinary kriging density estimates were checked by visually comparing specific gravity composites to block model estimates for sub-zones 5 and 100 as tests of the overall estimation. As well average specific gravity estimates for ordinary kriging were compared with global kriging for the two sub-zones.

Deleterious element estimates were checked by comparing composites to block model estimates. The main assumption made in estimating these elements is that if the U_3O_8 estimates are valid the deleterious element estimates are also valid as far as indicating that the concentrations of these elements will likely not affect the economic potential of the deposit.

Based on the above validation procedures, SRK is believes that the Roughrider West Zone block model is a fair and appropriate representation of the global mineral resources for the deposit with the current level of sampling.

13.12.2 Roughrider East Zone

Estimates were verified by conducting checks on sub-zone 2. Verification procedures included visual examination of block grades to drillhole composites, and comparing estimated grades at zero cut-off to nearest neighbour estimates, declustered means for each sub-zone and swath plots for sub-zone 3. All validation checks confirm that the block model is an appropriate estimate of the global tonnage and grade for the East Zone.

Deleterious element estimates were checked by comparing composites to block model estimates. The main assumption made in estimating these elements is that if the U_3O_8 estimates are valid the deleterious element estimates are also valid as far as indicating that the concentrations of these elements will likely not affect the economic potential of the deposit.

Based on the above validation procedures, SRK believes that the Roughrider East Zone block model is a fair and appropriate representation of the global uranium oxide mineral resources with the current level of sampling.

13.13 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

"a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

13.13.1 Roughrider West Zone

For the Roughrider West Zone, a potentially economic cut-off grade was determined by developing conceptual open pit shell using Whittle pit optimization software and reasonable optimization parameters based on discussions with Hathor and benchmarking with similar projects. This was done based on the initial thought that this zone would be mined by open pit methods and remains a viable option. Subsequent to the resource estimation, it was determined that underground ("UG") mining would be the preferred extraction methodology.

The following parameters were used for an open pit extraction scenario:

- U_3O_8 price US$80.00 per pound;
- Mining dilution and loss 5.0%;
- Average pit slope 39º;
- Mining costs US$2.00 per tonne;
- Toll milling, general and administration and transport costs US$90 per tonne; and
- Process recovery of 98%.

The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Roughrider Project. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

SRK considers that those blocks located above the maximum depth of the conceptual pit shell (200 metres above mean sea level) can be reported as a mineral resource at a cut-off grade of 0.05% U_3O_8. SRK notes that less than 1% of estimated blocks are located below that elevation.

The uranium mineralization at the Roughrider Uranium Deposit may extend into adjacent claims controlled by Fission Energy Corp. The Mineral Resource Statement prepared by SRK includes resource blocks above the 200m elevation and located only within the boundaries of the Roughrider Project controlled by Hathor, as determined using a property map provided by Hathor. SRK did not verify the legal validity of that boundary.

The conceptual open pit crosses the boundary with the Fission Energy Corp. Claims. This factor is not significant to the declaration of a mineral resource statement because mining regulations in Saskatchewan do not allow claim holders of adjacent properties to stop or restrict the development of mining projects. In fact the development of the project could benefit both companies.

Mineral Resources for the Roughrider West Zone have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines and are classified according to the “CIM Standards on Mineral Resources and Reserves: Definition and Guidelines” (December, 2005). The Mineral Resource Statement was prepared by G. David Keller, P.Geo (APGO#1235), an “independent qualified person” as this term is defined in National Instrument 43-101.

At a cut-off grade of 0.05% U_3O_8, the mineral resources for the Roughrider West Zone are estimated at 394,200 tonnes grading an average of 1.98% U_3O_8 in the Indicated category and 43,600 tonnes grading an average of 11.03% U_3O_8 in the Inferred category.

The Mineral Resource Statement for the Roughrider Uranium Deposit is presented in Table 13.8. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserve. SRK is unaware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that may materially affect the mineral resources. The effective date of the Mineral Resource Statement is November 29, 2010.

13.13.2 Roughrider East Zone

The "reasonable prospects for economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recovery. SRK considers that the Roughrider East Zone is amenable for underground extraction. To assist with determining a reasonable reporting cut-off grade, SRK considered the following assumptions:

- U_3O_8 price of US$80.00 per pound; and
- Process recovery of 98%.

Based on those assumptions, SRK considers that resource blocks above a grade of 0.4% U_3O_8 shows reasonable prospect for economic extraction from an underground mine and therefore can be reported as a mineral resource.

Potentially deleterious elements including arsenic, cobalt, copper, molybdenum, selenium and nickel are at low concentration and do not affect the reasonable prospects for economic extraction for this deposit.

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The mineral resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

The Mineral Resource Statement presented in Table 13.8 was prepared by G. David Keller, P.Geo. (APGO#1235) and Sébastien Bernier P.Geo. (APGO#1847) both “independent qualified persons” as this term is defined in National Instrument 43-101. The effective date of the Mineral Resource Statement is May 6, 2011.

Table 13.8: Mineral Resource Statement* for the Roughrider West Zone Deposit, Saskatchewan, SRK Consulting Inc., November 29, 2010 (West Zone) and, May 6, 2011 (East Zone)

Category	Quantity	Grade							Contained
	[Tonnes]	U_3O_8 [%]	As [%]	Co [%]	Cu [%]	Mo [%]	Ni [%]	Se [ppm]	U_3O_8 [lb]
West Zone									
Indicated High Grade Zone	58,200	10.68	0.17	0.03	0.41	0.22	0.15	46	13,703,000
Inferred High Grade Zone	36,600	13.07	0.69	0.10	0.57	0.26	0.55	56	10,546,000
Indicated Low Grade Zone	336,000	0.48	0.00	0.00	0.00	0.00	0.00	8	3,556,000
Inferred Low Grade Zone	7,000	0.31	0.00	0.00	0.00	0.00	0.00	4	48,000
Total Indicated West Zone	**394,200**	**1.98**	**0.03**	**0.00**	**0.06**	**0.03**	**0.02**	**13**	**17,207,000**
Total Inferred West Zone	**43,600**	**11.03**	**0.58**	**0.08**	**0.48**	**0.22**	**0.47**	**48**	**10,602,000**
East Zone									
Inferred Zone 1	26,000	12.17	0.02	0.01	1.49	0.05	0.01	15	6,970,000
Inferred Zone 2	30,000	13.34	0.03	0.01	1.34	0.13	0.02	49	8,930,000
Inferred Zone 3	32,000	17.39	0.03	0.01	0.15	0.14	0.02	22	12,140,000
Inferred Zone 4	3,000	1.34	0.00	0.00	0.08	0.03	0.00	9	80,000
Inferred Zone 5	11,000	1.65	0.01	0.01	0.40	0.14	0.01	14	390,000
Inferred Zone 6	12,000	3.57	0.01	0.01	1.05	0.06	0.01	31	940,000
Inferred Zone 7	5,000	6.84	0.00	0.00	0.04	0.03	0.00	13	680,000
Total East Zone Inferred	**118,000**	**11.58**	**0.02**	**0.01**	**0.86**	**0.10**	**0.02**	**27**	**30,130,000**
Combined East and West Zones									
Total Indicated	**394,200**	**1.98**	**0.03**	**0.00**	**0.06**	**0.03**	**0.02**	**13**	**17,207,000**
Total Inferred	**161,600**	**11.43**	**0.17**	**0.03**	**0.76**	**0.14**	**0.14**	**32**	**40,730,000**

*CIM Definition Standards have been followed for classification of mineral resources. The cut-off grade of 0.05 percent U_3O_8 for the West Zone and 0.4 percent U3O8 was for the East zone. U_3O_8 price of US$80 per pound and assumed. Reasonable prospect for economic extraction" assumes open pit extraction for West Zone and underground extraction for East Zone and metallurgical recovery of 98 percent. Mineral resources are not mineral reserves and do not have demonstrated economic viability. Totals may not add correctly due to rounding.

13.14 Previous Mineral Resource Estimates

Mineral resources for the Roughrider Uranium Deposit were previously estimated by Scott Wilson RPA Inc. ("RPA") as documented in a technical report dated December 18, 2009 (Table 13.9). Using a cut-off grade of 0.06% U_3O_8 indicated resources were reported at 116,000 tonnes at 2.57% U_3O_8 and Inferred resources were reported at 83,000 tonnes at 3.00% U_3O_8.

Differences between the Scott Wilson RPA 2009 resource model and the one prepared by SRK and reported herein are largely due to additional drilling completed by Hathor on the project.

More closely spaced drilling allows the delineation of high and low grade domains with a higher level of confidence.

Table 13.9: Mineral Resource Statement* for the Roughrider Uranium Deposit, Saskatchewan, Scott Wilson RPA, September 1, 2009

Category	Quantity (Tonnes)	Grade							Contained
		U_3O_8	As	Co	Se	Cu	Mo	Ni	U_3O_8
		(%)	(%)	(%)	(ppm)	(%)	(%)	(%)	(lb)
Indicated High Grade Zone	10,000	17.28	0.69	0.07	47	0.38	0.21	0.55	3,850,000
Inferred High Grade Zone	7,000	19.15	0.63	0.05	60	0.46	0.25	0.58	3,050,000
Indicated Outer Zone	106,000	1.17	0.13	0.03	7	0.22	0.10	0.10	2,730,000
Inferred Outer Zone	75,000	1.46	0.10	0.03	10	0.16	0.09	0.08	2,420,000
Indicated Total	**116,000**	**2.57**	**0.17**	**0.04**	**10**	**0.23**	**0.11**	**0.14**	**6,580,000**
Inferred Total	**83,000**	**3.00**	**0.15**	**0.03**	**14**	**0.18**	**0.10**	**0.13**	**5,470,000**

Notes:
1. CIM Definition Standards have been followed for classification of Mineral Resources.
2. The cut-off grade of 0.06% U3O8 was estimated using a U3O8 price of US$65/lb and assumed operating costs and recoveries.
3. High U3O8 grades were cut to 30% in the outer zones, and 50% in the high grade zones.
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
5. Totals may not add correctly due to rounding.

13.15 Sensitivity Analysis

The block models constructed by SRK for the two zones shows that U_3O_8 grades and quantities estimates are sensitive to the selection of cut-off grades.

As sensitivity analysis the global quantities and grade estimates in the entire block model are presented at different cut-off grades as grade and tonnage curves for the West Zone in Figure 13.3 the East Zone in Figure 13.4.

The reader is cautioned that figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grades.

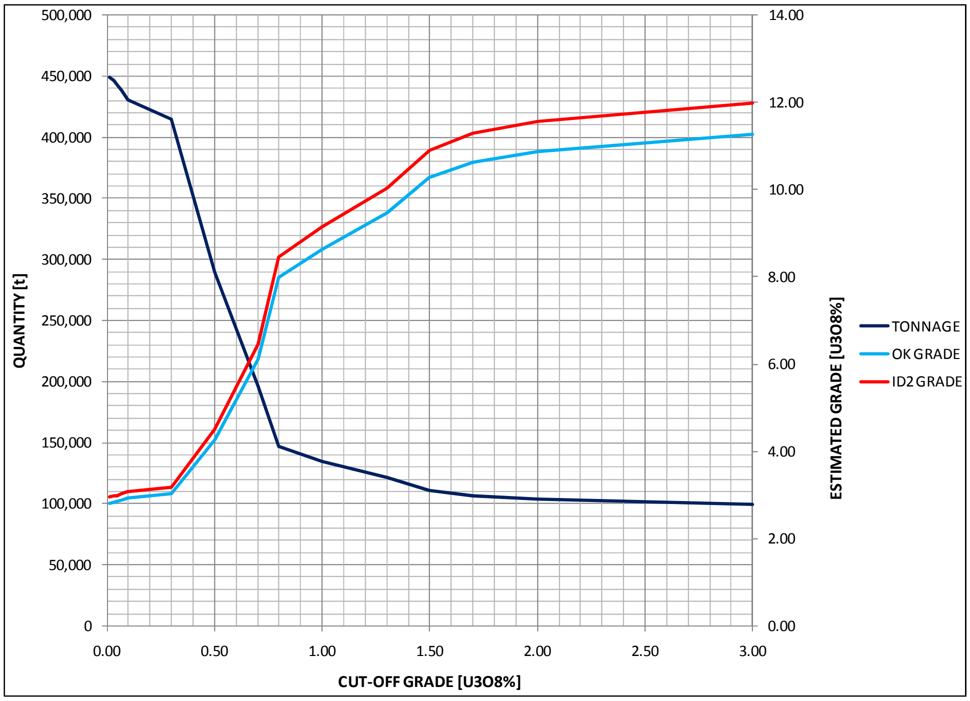


Figure 13.3: Global Tonnage and Grade Curve for West Zone

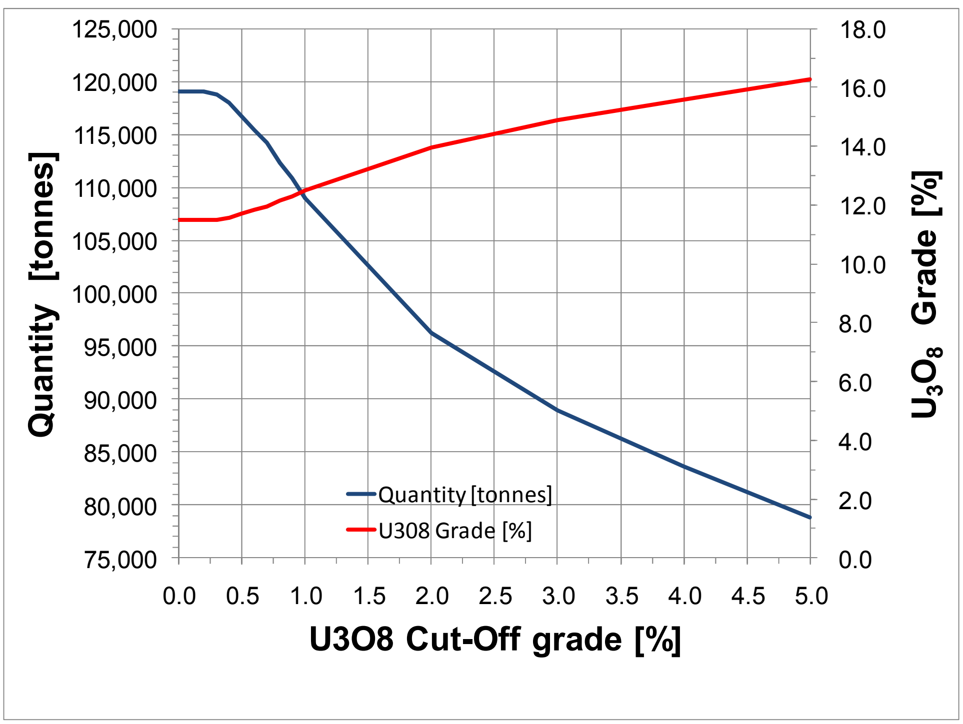


Figure 13.4: Global Tonnage and Grade Curve for the Roughrider East Zone

14 Mineral Reserve Estimates

No mineral reserves can be declared for the Roughrider property as a preliminary feasibility study has not yet been conducted.

15 Mining Methods

15.1 Mining Context

The Roughrider deposit is located predominantly in basement rock-hosted uranium mineralization. Mineralization occurs in pods and semi-massive veins and semi-massive replacements of uraninite.

Exploration drilling has identified three mineralized zones at the Roughrider deposit: the West, East and Far East. Only two mineralized zones were evaluated in this study: the West zone and the East zone. An evaluation of the Far East Zone was not considered at this time as the zone is currently undergoing its first mineral resource estimate.

The main geological and geotechnical contextual considerations for the Roughrider deposit are:

- Deposit Geometry and Mineralization:

  The East & West Roughrider deposits lay beneath about 200 m of sandstone of variable geotechnical quality.

  The West Zone has a strike length of approximately 200 m with an across strike extent of 90 m. Uranium mineralization occurs at depths of 200 m to 280 m below surface and is hosted predominantly within basement rocks. Only minor amounts of uranium occur at or above the unconformity. Uranium mineralization is highly variable in thickness and style.

  The East Zone has a surface projection of approximately 105 m long in a north-easterly direction, which corresponds to a down-dip length of approximately 125 m, and an across-strike extent about 80 m. Uranium mineralization has a vertical extent of up to eighty to 100 m, starting at depth approximately 240 m from surface, and some 30 m to 50 m below the unconformity. This is slightly deeper than the Roughrider West Zone.

- Mineralized Material Grades:

  The high grade mineralization of the West zone contains about 98,000 t with an average grade of 11.6 % U_3O_8 and is surrounded by low grade mineralization containing about 343,000 t with an average grade of 0.5 % U_3O_8.

  More than 90% of the East zone mineralization is represented by high grade material with average grade of 12.5% U_3O_8.

- Geotechnical Characteristics:

  No dedicated geotechnical drilling and analysis have been conducted on the West and East zones The assumed rock mass condition assumptions were s based on core photographs, core inspection and core logs as described in Section 23.1.

- Groundwater:

  Hydrogeology testing has not been conducted on the West and East zones or in the surrounding area but it was assumed that mining would be impacted by groundwater located in the overlying sandstones, at the unconformity and in fractures in the basement rock. As a result, water inflow mitigation measures would be required including a fully supported mining method, ground freezing and grouting.

- Radiation

  Due to the high grade of the Roughrider mineralized zones, radiation protection would impact UG ventilation systems, mine design and mine operations.

15.2 Mining Method Selection

The main contextual factors for determining an appropriate mining method for the Roughrider deposit were: radiation from very high grade uranium, water bearing sandstones, the unconformity overlaying the mineralized basement rocks, weak rock formations, the irregular geometry of the mineralization, high value of the mineralized material and the 250 m depth of the deposits.

SRK determined that Roughrider deposits would be mined by underground methods due to the depths of the deposits and the location of the West deposit under South McMahon Lake. The selected mining method would need to minimize the hazards of high radiation exposure and be compatible with groundwater control methods. It would also need to be selective in mining waste, high grade and low grade mineralization, and be able to support high mining extraction factors.

The raisebore mining method was selected to be appropriate to satisfy the requirements described above.

To minimize the risk of any potential water ingress from the overlying sandstones and the unconformity, it was assumed that ground freezing would be provided to establish a freeze-wall that umbrellas the deposits prior to mining. Ventilation raises would also be developed inside freeze walls.

15.3 Mining Method Description

The raisebore method has been used elsewhere in the Athabasca Basin and has proven to be successful in achieving satisfactory safety and production levels.

No men would be exposed to radiation or rock fall from freshly blasted ground. Production raiseboring would be done from a well-supported raise chamber.

The raisebore mining method is started with the development of a raise chamber level located above the mineralized zone and development of an extraction or mucking drift below the mineralized zone. The raisebore chamber level serves as the location for the raiseboring machines. The extraction or mucking level provides access to the bottom of the raisebore hole to allow for the mucking of cuttings. A 311 mm diameter pilot hole would be drilled with a roller bit from the raise chamber through the ore zones to the mucking level below. When the pilot hole breaks through into the mucking level, the roller bit would be removed and replaced with a reaming head. The pilot hole then would be reamed to a 3.05 m diameter from the bottom up. The raises would be drilled vertically, or at an angle not less than 75°.

A preliminary radiometric survey of the pilot hole would be required to make initial planning for ore and waste extraction. Based on radiometric measurements it could be established how much initial waste material would be produced from pilot hole reaming, and when the reaming would intersect low grade and high grade material. The limit of the raise reaming would be reached when mineralization fell below the economical cut-off.

The cuttings from the reaming would fall by gravity to the mucking level where they would be mucked out by remote controlled Load-haul dump loader ("LHD"). The waste from the initial reaming and from between mineralized zones, in some cases, would be mucked directly to the trucks and hauled to the backfill plant to be used for backfill of mined-out raises. The low grade and high grade material would be mucked to the dedicated ore passes. A radiometric scanning system would be used to determine the grade of mucked material. Additional scanning would be done at the bottom of the ore passes to ensure appropriate blending of high grade and low grade material to provide required for the mill feed grade of 3.3% U_3O_8.

When the reaming of the raise is complete, the reaming head would be lowered to the bottom of the raise and detached. The raiseborer would be relocated to the next location to drill a new raise.

The mined-out raise would be backfilled with cemented rock fill to permit the mining of the next raise in sequence.

The raise spacing layout would provide raises overlap to maximize extraction of mineralization. A provision of 15% dilution from backfill material was assumed in the mining inventory estimate to account for raise intersections with mined-out and backfilled raises.

When raise mining is complete below the raise bore chamber, the chamber would be backfilled. A new raisebore chamber would be developed adjacent to the previous one, intersecting the backfilled chamber. The same sequence would occur on the mucking level.

A typical raiseborer set-up is shown in Figure 15.1 below.

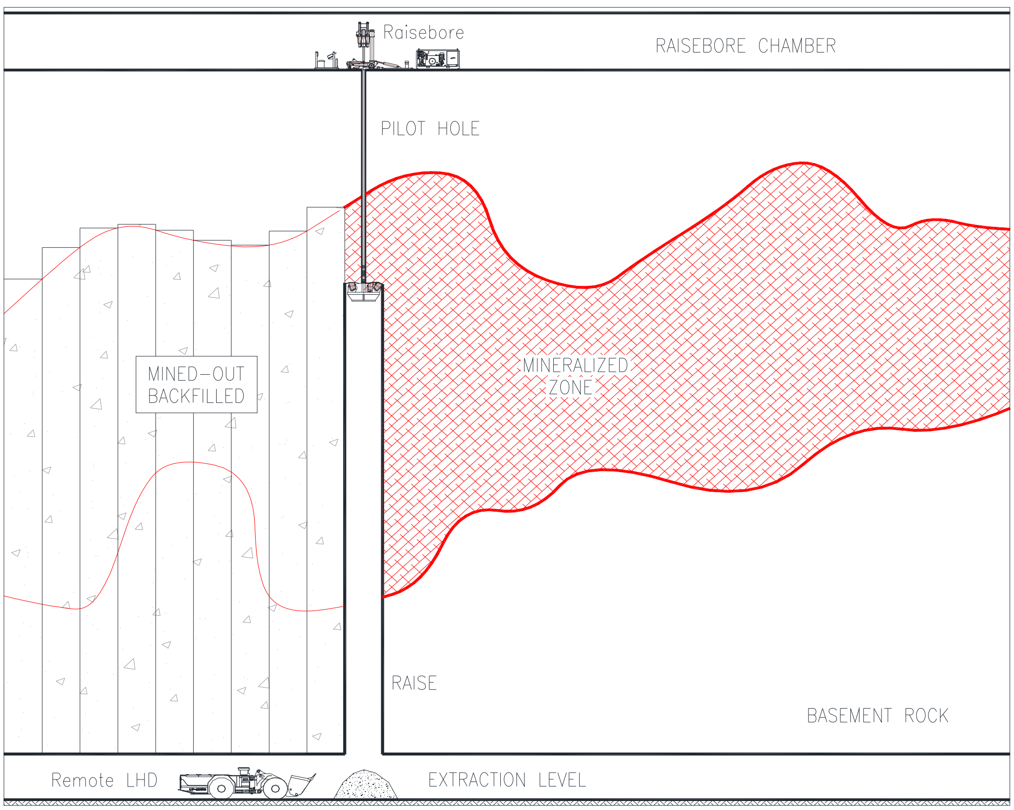


Figure 15.1: Longsection View of Raisebore Mining (SRK)

15.4 Mining Inventory

The summary of the mining inventory for the proposed life of mine ("LOM") plan is shown in Table 15.1. The East zone contains only inferred resources while the West zone contains a combination of indicated and inferred resources.

Table 15.1: **Preliminary Roughrider Mining Inventory Summary**

Zone	Classification	Diluted Tonnes (t)	Cut-off Grade (U_3O_8%)	Diluted Grade (U_3O_8%)	Contained Metal (Mlb U_3O_8)
West	Indicated	286,000	1.06	2.5	15.8
	Inferred	58,000	1.06	7.2	9.2
East	Inferred	388,000	1.06	3.3	28.2

*These preliminary assumptions were later revised with more detailed/accurate estimates

15.4.1 Cut-off Criteria

The mine design for the West and East zones of the Roughrider deposit was initiated with the development of input parameters to estimate the cut-off grade. These parameters included estimates of metal price (US$60 /lb U_3O_8), exchange rate, mill recovery, royalties, on-site operating costs, and mining dilution.

A preliminary estimate of the total on-site costs of $960 which included mining operating cost of $425 /t, processing cost of $410 /t, and G&A cost of $125 /t, were used to determine a cut-off grade.

Table 15.2 outlines the preliminary input parameters used in the cut-off grade ("COG") calculation. More detailed cost estimation work was performed after the mining shapes were determined. The difference between the preliminary cost estimates and subsequent, more detailed, cost estimates were not deemed sufficiently different to modify the COG or mining shapes.

Table 15.2: Cut-Off Grade Calculation

Parameter	Unit	Value
Metal Recovery		
U_3O_8 Price	\$US/lb U_3O_8	60
Exchange Rate	\$C/\$US	1.05
U_3O_8 Price	\$C/lb U_3O_8	63.16
Payable Metal	% U_3O_8	100
Process Recovery	%	97.5*
Refining/Freight/Insurance/ Marketing	\$C/lb U_3O_8	N/A
Royalties @ 5% NSR	\$C/lb U_3O_8	3.08*
Net U_3O_8 price	\$C/lb U_3O_8	58.5
OPEX Estimates		
Mining Cost	\$ /t milled	425*
Processing Cost	\$ /t milled	410*
G&A cost	\$ /t milled	125*
Total Site Cost	\$ /t milled	960*
Cut-off Grade		
Plant feed Cut-off Grade	% U_3O_8	0.75
Dilution	%	30
In-situ Cut-off Grade	% U_3O_8	1.06

A 1.06% U_3O_8 cut-off grade was applied for mining inventory estimates.

15.4.2 Dilution and Recovery

Dilution was envisioned to come from the following sources:

- Dilution from the low grade mineralized material with a below economical cut-off grade of 1.06% U_3O_8;
- Waste dilution from production raiseboring drilling;
- Dilution from backfill material;
- Additional waste dilution for the blending of mineralized material to the designed mill feed grade of 3.3% U_3O_8.

Dilution was defined as the ratio of waste, or as low grade material to high grade mineralized material. It was estimated that about 15% dilution would be produced from the backfill material and another 15% from the waste surrounded the mineralized zones.

The total production dilution of 30% with a zero grade (0% U_3O_8) was assumed for the mining inventory estimates.

A 97% recovery factor was applied to each individual mining block to account for production losses due to drillhole deviation and the possible inability to complete the reaming of some pilot holes.

15.4.3 Mining Inventory Estimation

The Gemcom / Surpac mine planning software was used for the analysis of the mineral deposit geometry and grades. The economical cut-off grade of 1.06% U_3O_8 was used as a basis for the mining inventory estimation.

The following assumptions were used for the mining inventory estimates:

- The production raises would be reamed from the mucking level to the top of the mineralized zone. Raise reaming would be limited to the grade of mineralization below the economical cut-off grade of 1.06% U_3O_8;
- Waste material at the bottom of the production raise and between the zones of mineralization could be mined separately and used as backfill material;
- Mineralized material with grades below 1.06% U_3O_8 would be considered as incremental and would be mucked to the low grade ore pass and would be used for the blending of the high grade material to produce the required 3.3% U_3O_8 for the mill feed;
- 97% extraction for raiseboring mining method;
- 30% external dilution with zero grade due to waste dilution and dilution from backfill.

The tonnages and grades were extracted from the block model. The tonnage and grade of vertical columns of blocks from the mucking level elevation to the top limit of the proposed raise extraction was estimated. The amount of waste inside the column was then estimated. If the associated value of the column would cover mining and processing costs of the mineralized material and mining of waste material (as waste material would not be processed), the column was included in the mining inventory. If the combined costs of the column were estimated to be higher than value of mineralization inside the column, the column was rejected from the mining inventory.

The material above and below the COG was estimated separately to insure if the average grade would satisfy mill feed requirements after applying 30% of waste dilution. If diluted grade was estimated to be lower than 3.3% U_3O_8, some of low grade material would be used as a backfill and not to be processed. A 97% recovery factor and 30% dilution were applied to the estimated tonnage and grades.

If the diluted grade was higher than 3.3% U_3O_8, an additional waste dilution was applied to produce required for the mill feed grade.

The mining inventory estimate is shown in Table 15.3.

Table 15.3: Roughrider Mining Inventory Estimate

Zone	Classification	Description	Tonnes (t)	Grade (U_3O_8%)	Contained Metal (Mlb U_3O_8)
West	Indicated	Above 1.06%	90,950	7.75	15.54
		Below 1.06%	129,210	0.24	0.69
		Sub-total	**220,160**	**3.34**	**16.23**
		Recovered (97%)	213,560	3.34	15.74
		Dilution with 0% grade	72,350	0	0
		Total Diluted	**285,910**	**2.5**	**15.74**
	Inferred	Above 1.06%	41,310	10.5	9.57
		Below 1.06%	3,570	0.26	0.02
		Sub-total	**44,880**	**9.69**	**9.59**
		Recovered (97%)	43,540	9.69	9.3
		Dilution with 0% grade	14,750	0	0
		Total Diluted	**58,290**	**7.24**	**9.3**
East	Inferred	Above 1.06%	96,040	13.69	28.98
		Below 1.06%	7,760	0.65	0.11
		Sub-total	**103,800**	**12.71**	**29.09**
		Recovered (97%)	100,680	12.71	28.22
		Dilution with 0% grade	287,220	0	0
		Total Diluted	**387,900**	**3.3**	**28.22**

15.5 Conceptual Mine Design

It was assumed that the sandstones and unconformity are water bearing and can only be developed with grout or freeze cover. Although most of the mineralized zones are located in basement granites, they are close enough to the unconformity (0-30 m) that the impact of any potential water ingress from the overlying sandstones and unconformity is planned to be mitigated by establishing a freeze-wall that umbrellas the deposits. Ventilation raises would also be developed inside freeze walls.

The main access into the mine was planned to be provided via a decline developed under a grout cover. The decline was planned to be developed in the sandstones in an area with the best geotechnical and hydrogeological characteristics and pass through the unconformity in an area of good rock quality based on drilling results.

Two ventilation raises would be developed to provide a primary ventilation circuit and second egress from the mine. Internal ventilation raises would be developed to provide flow through ventilation for the West zone freeze drifts and provide direct route for contaminated exhaust air.

Freezing drifts would be developed on perimeter of the mineralized zones to establish freeze walls around the planned production areas.

Raisebore chambers would be developed 8 m to 12 m above mineralized zones and mucking or production levels at 5 m to10 m below mineralized zones.

The ore pass system would be comprised of a low grade ore pass and a high grade ore pass to provide storage of production material. The two ore passes would be utilized to blend high grade and low grade material to produce the desired grade for the mill feed. The storage capacity of the ore pass system would be a minimum of one day of mine production.

The conceptual mine design is shown in Figures 15.2 and 15.3.

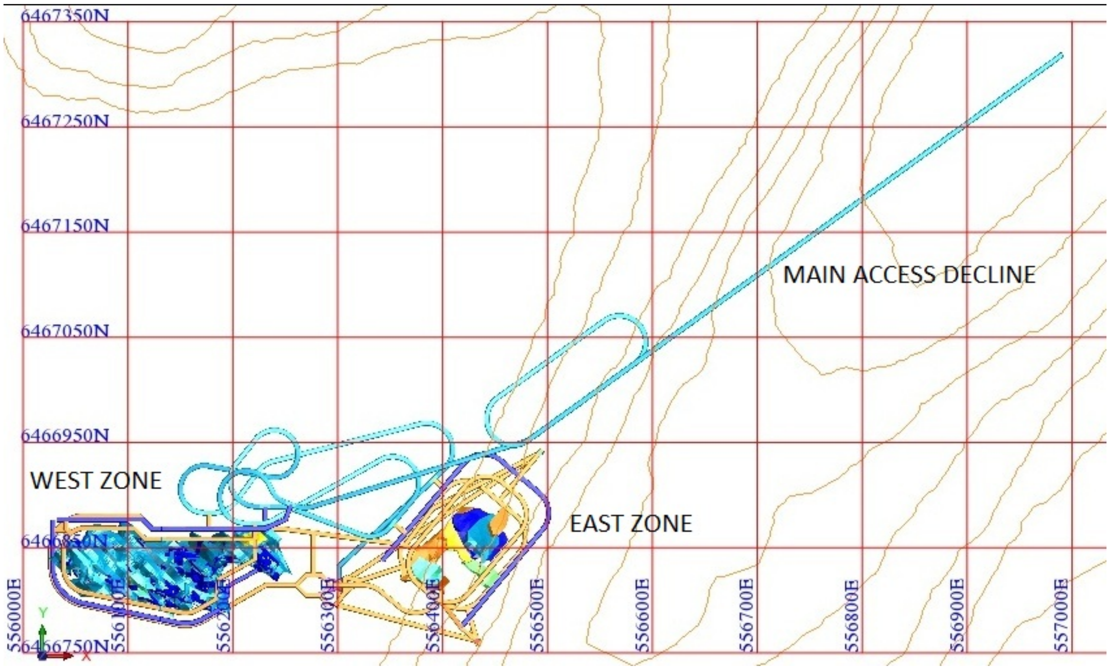


Figure 15.2: Roughrider Conceptual Mine Design. Plan View

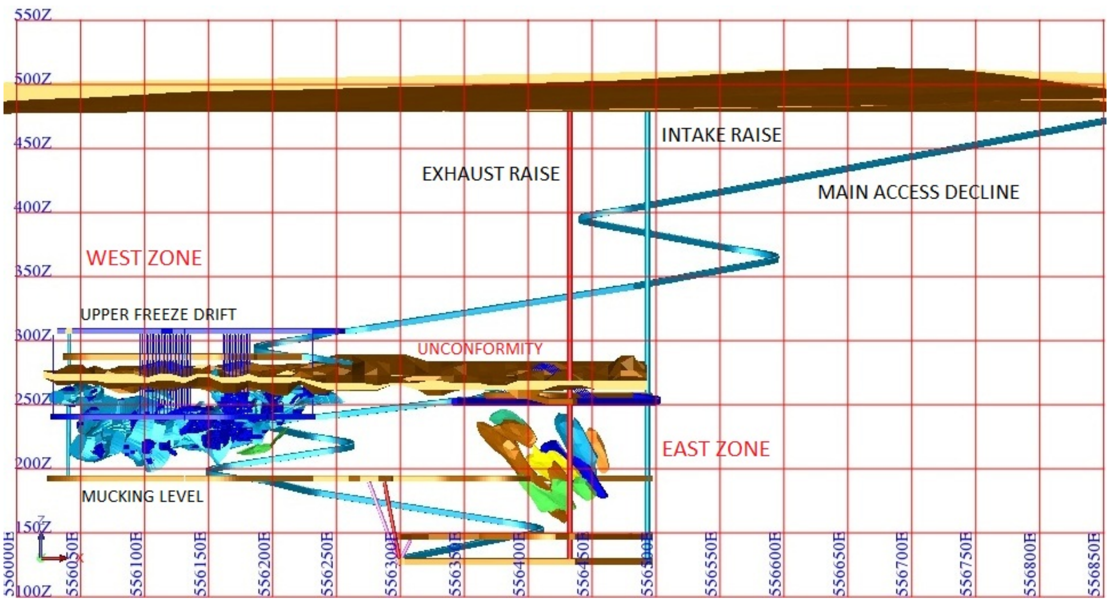


Figure 15.3: Roughrider Conceptual Mine Design. Section View

15.5.1 Mine Access

Two possible methods of mine access were considered in the study: vertical shaft and decline access. It was assumed that development of the vertical shafts would require ground freezing from surface to about 20 m below the unconformity. The decline development was assumed to be done under a continuous full grout cover.

A trade-off study was performed to provide economical assessment of those two options of mine access. The following assumptions were made to compare two options of the mine access:

- The shaft access option would require development of two shafts: production and ventilation;
- The decline access option would require development of decline and two ventilation raises: intake and exhaust;
- The shafts and ventilation raises need to be frozen from the surface to a depth of about 20 m below the unconformity;
- The access decline would be developed with grouting to the same depth as shaft sinking with freezing;
- The shafts would be developed by sinking and the raises by means of raiseboring;
- The decline access option would not require any special equipment installed after development is completed while the shaft option would require a hoisting system to be installed and shaft infrastructure to be developed (shaft stations, loading pockets, etc.);
- An access needs to be provided to the freezing drifts, drilling and mucking levels in both options;
- The production development would be the same for both options.

The cost of development and schedule were compared for two options. Providing an access to the freezing drifts was assumed as a critical path, because ground freezing around the mineralized zones would take about nine months (during that time the most of development required to start production would be completed as it would be in base ground). It was estimated that the access to the freezing drifts would be provided two months earlier by shaft development, but the shaft option would be over $10 M more expensive than the decline option.

The access to the mineralized zones via 15% gradient decline was selected as the most economical method based on the results of the trade-off study. Development of two ventilation raises: intake and exhaust, using ground freezing would be required for that option.

More detailed investigation of ground conditions and hydrogeology would be required to confirm if decline could be developed with grouting and to select an appropriate grouting method and grouting parameters. If such investigation would confirm that decline development with grouting would not be possible, the shaft sinking with ground freezing would be done instead of raise development proposed for the decline option.

One of the ventilation raises could become a production shaft and another could be used to provide ventilation circuit and a second egress.

The size of the decline was selected according to the mobile equipment size, required clearances, and ventilation requirements during development and production. It was estimated that a 5.0 m wide by 5.0 m high decline would be satisfactory for a 30 t truck and ventilation requirements. A 25 m ramp curve radius was assumed for the convenience to drive a mobile drill jumbo.

Re-muck bays were planned to be developed every 150 m along the decline to allow efficient use of the drilling equipment and would hold two rounds of development muck. The re-muck bays would be of a similar size as the decline and would be typically 15 m long. After they are no longer used for development, the bays would be used for equipment storage, pump stations, refuge chambers, etc.

Installation of 2.4 m fully grouted resin rebar bolts on the back and the walls of the decline on 1.2 m x 1.2 m pattern, 100% mesh coverage and an allowance for 50 mm of shotcrete was assumed for decline ground support.

Two ventilation raises would be developed to provide a primary ventilation circuit. One of the ventilation raises located on the north side of the deposit, would be used for intake air. It would have a man-way equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency. Having two intake ventilation sources, main access decline and intake ventilation raise, would provide an opportunity for the main access to the mine and emergency egress on fresh air.

The second ventilation raise would provide exhaust from the mine. It would be located on the south side of the deposit that would provide a dedicated ventilation root for the exhaust air and eliminate any re-use of contaminated air.

The top 230 m portion of the ventilation raises from the surface to about 25 m below unconformity was planned to be developed using ground freezing. The raiseborer would then be relocated underground and the remaining 125 m of the raise would be developed to the bottom of the mine without freezing.

Access to the drilling and mucking levels would be through the crosscuts developed from the main decline. The ventilation drifts and crosscuts would be developed to connect the level development to the ventilation raises.

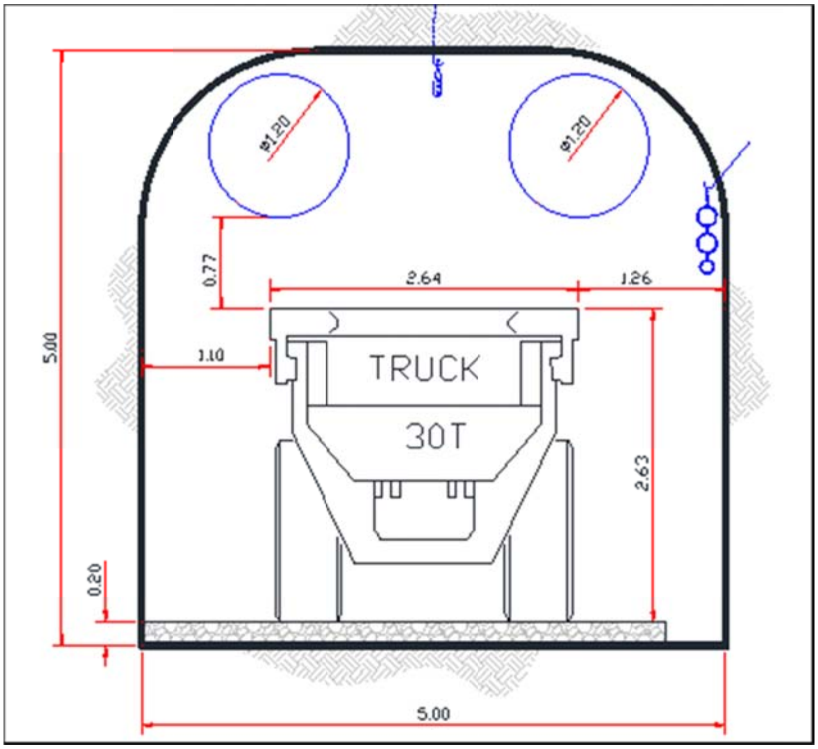


Figure 15.4: Proposed Decline Cross Section

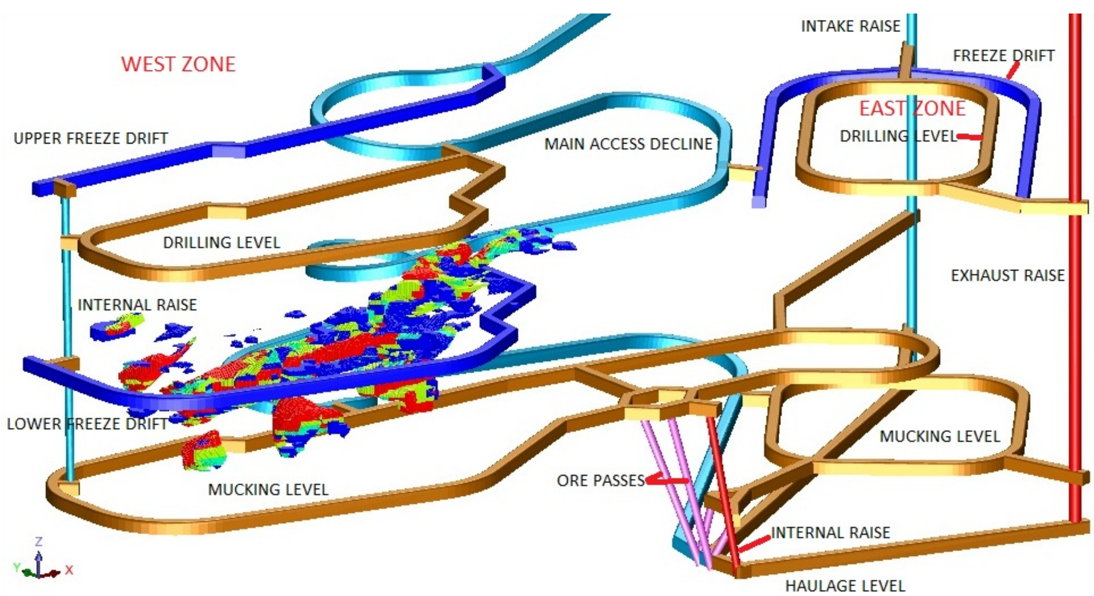


Figure 15.5: Roughrider Mine Access Development. Isometric View

15.5.2 Ground Freezing

Ground freezing around the mineralized zones would be required prior to drilling level development and production. The freeze walls would be established to isolate production areas from the water-bearing sandstones and the unconformity.

Ground freezing would be accomplished by drilling a series of holes from the dedicated freezing drifts developed on the perimeter of the mineralized zones and circulating -35°C calcium chloride brine through the rock until frozen. The holes would be drilled at a 3.0 m spacing. Freeze pipes would be installed inside the holes and connected to intake and return lines from the freeze plant located on surface.

Approximately nine months would be required for freezing the walls. The walls would need to be completely enclosed to isolate the proposed mining areas from water-bearing ground. The freeze hole drilling and freezing periods were considered in the mine schedule prior to production.

The East Zone, located below unconformity, would be isolated from the water-bearing ground by creating two cover walls above mining area. The freeze holes would be drilled from the opposite sides of the mineralization to provide a shelter type of isolation (see Figure 15.6).

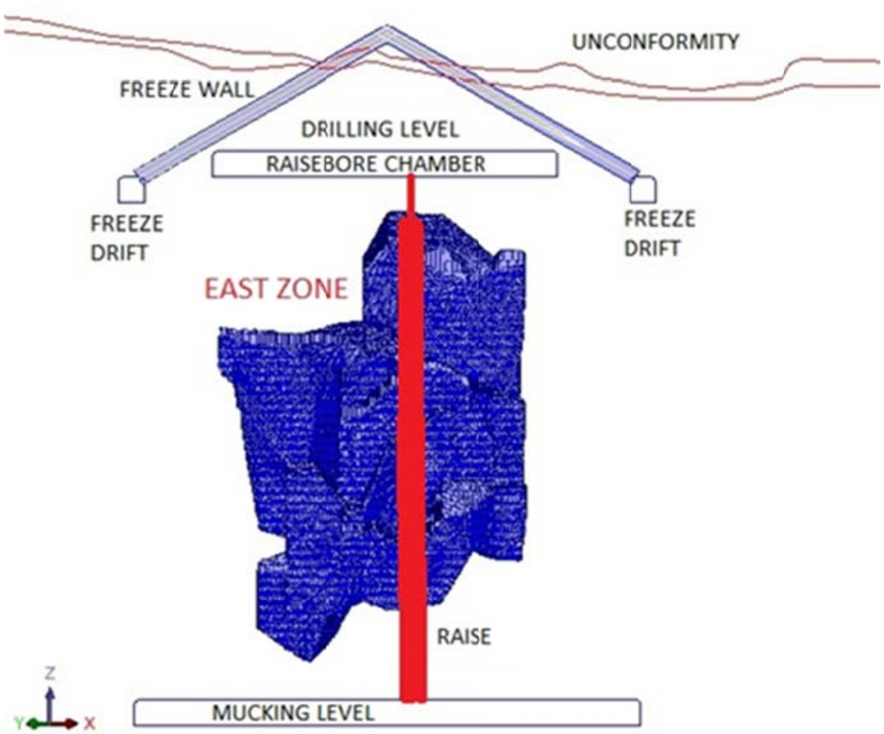


Figure 15.6: East Zone Freezing

A different arrangement of the freeze walls would be done in the West Zone, as a portion of mineralization is located above unconformity. Two freezing drifts would be developed at different elevations: one – approximately 25 to 30 m above unconformity in sandstone and another about 25 m below unconformity in the basement rock. The horizontal freeze holes would be drilled from the upper drift across the West Zone and vertical holes would be drilled down along one side of the zone. The freeze holes from the lower drift would be drilled vertically up around the perimeter of the zone to enclose the freezing area (see Figure 15.7).

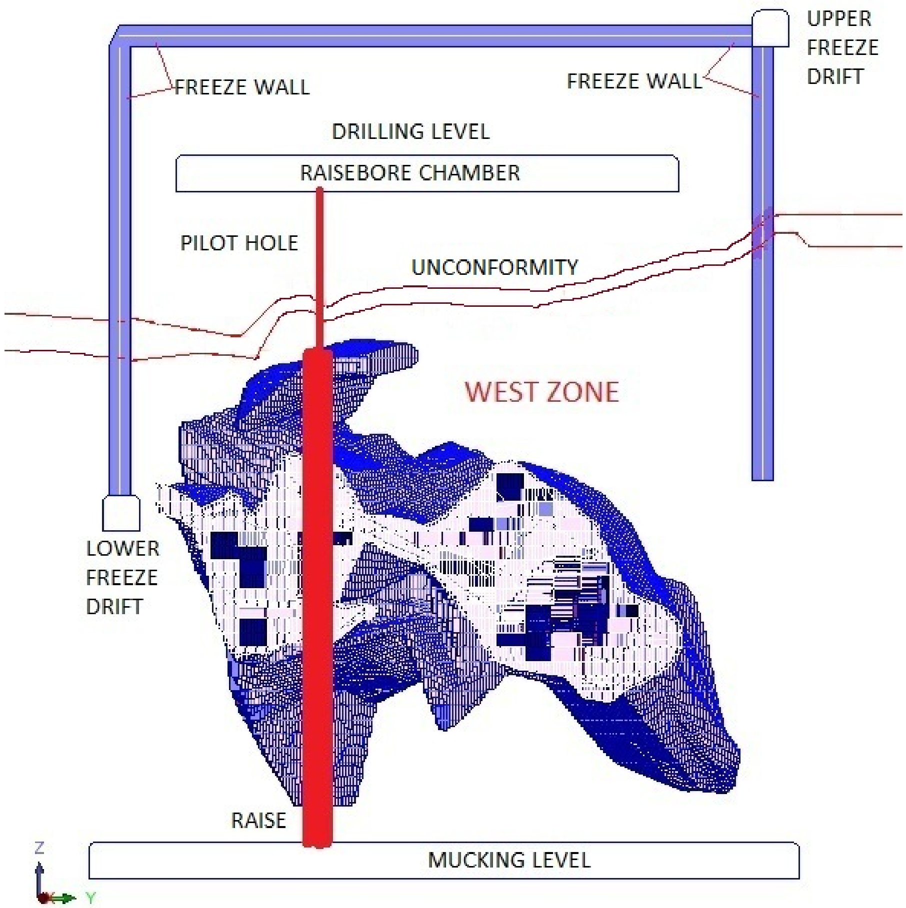


Figure 15.7: West Zone Freezing

15.5.3 Haulage

The waste rock from the development headings is planned to be mucked by LHDs directly to the trucks or to remuck bays located up to 150 m from the face. The waste rock would be hauled by the 30 t trucks to the waste dump on surface during the pre-production period. When underground mine production commences, it is proposed to use mine waste rock from development and production drilling as backfill. The same trucks would carry backfill material from the surface to an UG backfill plant, blended with cement and placed in mined-out raises.

The mineralized material from production raiseboring is planned to be mucked by LHDs with remote control and loaded directly onto ore passes. 5.4 m^3 LHDs were selected for mine production. The same size of LHD would be used for mucking of development waste and could be used for production when required.

Covered UG haul trucks would be loaded at the bottom of the ore passes and carry mineralized material from the mine directly to the mill. A slurry pumping system could be considered as an alternative method of material handling.

It was estimated that one truck per shift would be required to haul 200 t/d of mineralized material from underground to the mill and one truck per shift for waste haulage.

15.5.4 Backfill

The raiseboring mining method is designed to use cemented rock fill (“CRF”). CRF consists of waste rock mixed with cement slurry to improve the bond strength between the rock fragments. Waste rock would be sourced from mine development and from production raiseboring. Using waste rock as a backfill material would reduce the environmental impact on surface.

Backfill preparation would occur underground in a backfill chamber developed above mineralized zones. The rock would be crushed, screened, then mixed with cement slurry and placed in the production raises. Cement would be delivered underground as a dry product in bags or totes.

One of the advantages of CRF would be a high strength to cement content ratio. A cement content of 6% was assumed for CRF and used for the backfill cost estimation purpose.

The backfill test program would be required for future studies to assess waste properties and define the optimum cement binder content, backfill strength and backfill curing time to obtain required strength.

15.5.5 Mine Services

Ventilation

For the purpose of estimating ventilation requirements for the Roughrider underground operation it was assumed that adequate dilution of the exhaust produced by the underground diesel equipment fleet could be used as the design criteria and would sufficiently control for radon exposure in working areas. Exposure to possible higher concentrations of radon along with exposure to Long Live Radioactive Dust (“LLRD”) would be controlled through the mine and ventilation design to ensure the workers would always remain upwind of any potential sources.

Air volume was calculated on a factor of 0.06 m^3/s per utilised kW of diesel engine power. The kW rating of each piece of underground equipment was determined and then utilization factors, representing the diesel equipment in use at any time, applied to estimate the amount of air required. Ventilation losses of 25% and a safety factor of 50% were included for the total ventilation requirements. Table 15.4 lists the air requirements for full production with the total of a 150 m^3/s flow rate.

Table 15.4 : Ventilation Requirements at Full Production

Equipment Detail	Units	Quantity	Power Rating (kW)	Utilization	Factored Power (kW)	Air Volume Required (m^3/s)
Jumbo (2 boom)	ea.	1	74	50%	37	2.21
Rockbolter	ea.	1	55	50%	28	1.66
Load-Haul-Dump, 4.6 m^3	ea.	2	220	100%	440	26.4
Haulage Truck, 30 t	ea.	1	298	100%	298	17.9
Grader	ea.	1	149	100%	149	8.95
ANFO Loader	ea.	1	95	50%	48	2.86
Personnel Carrier	ea.	2	112	50%	112	6.71
Mechanics Truck	ea.	1	112	50%	56	3.36
Scissor Lift	ea.	1	112	75%	84	5.03
Supervisor/Engineering Vehicle	ea.	2	95	50%	95	5.73
Mec/Elec Vehicle - Scissor Lift or other	ea.	1	95	50%	48	2.86
Forklift/Tractor	ea.	1	63	50%	32	1.9
Total					1,426	86
Losses		25%				21
Safety factor		50%				43
Total Ventilation Requirements						**150**

The main exhaust fan would be installed on surface at the collar of the exhaust raise. An exhaust ventilation system is the most efficient method of diluting and removing radon progeny from the underground workings. Air would be drawn through the main decline and secondary intake raise from surface.

The ventilation system design was modeled using Ventsim Visual, an underground mine ventilation software program. All the airways were modeled to the dimensions of the mine design.

Minimum airflow requirements as shown in Table 15.5 were estimated for the various working areas in the mine, based upon the expected maximum equipment power to operate at one time in the area.

Table 15.5: Ventilation Requirements at Full Production

Working Areas	Expected Max Equipment	Air Volume (m^3/s)
Freeze Drifts (After Development)	1 Light Vehicle	5
Raisebore chamber	1 LHD x 50% safety factor	20
Mucking Level	(1 LHD + 1 Light Vehicle) x 50% safety factor	30

In addition to meeting the requirements of the various work areas listed in Table 15.5, a minimum airflow of approximately 35m^3/s was maintained on all sections of the main decline to support any development in the West Zone. The ventilation regulators, doors, and bulkheads that would be used to provide these airflows were modeled accordingly.

Fan and power requirements for the main fan were then estimated using the foreseen ventilation network at the start of full production in the East Zone as shown in Figure 15.8.

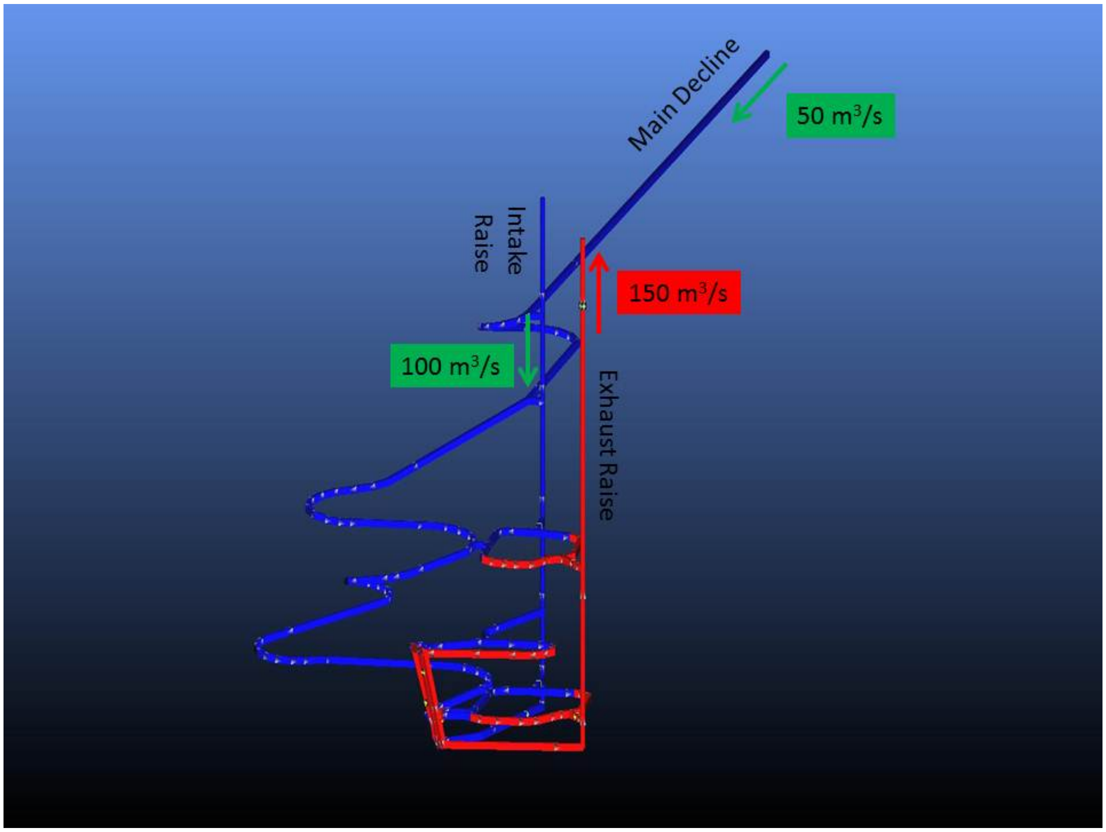


Figure 15.8: Ventilation Circuit at Start Production

Using a motor efficiency of 90% and a fan efficiency of 75%, the main fain would require approximately 70 kW of power to supply a total pressure of approximately 300Pa at 150m^3/s. Actual efficiencies and power requirements will depend on final fan selection.

Ventilation of Headings during Development

An air flow of 32m^3/s would be required to dilute and remove exhaust from a 30 t truck, a 5.4 m^3 LHD, and a two-boom jumbo working in a development heading.

Table 15.6: Ventilation Requirements for Development Heading

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m^3/s)
LHD, 5.4 m^3	1	220	100	220	13
Truck, 30 t	1	298	100	298	18
Jumbo, two-boom	1	111	10	11	1
Total					**32**

The requirements for auxiliary ventilation were estimated for the 1,800 m long development heading, as the longest decline development distance required by the mine development program. The auxiliary ventilation fans and ventilation ducts would be used to provide the required amount of air at the development face. Only the resistance of the duct was considered to calculate the pressure loss and power requirements, as the resistance of the heading is negligible by comparison. Two 75 kW auxiliary fans with two 1.2 m diameter ventilation ducts were estimated to be required for the decline development.

The 50 kW and 40 kW fans and 1.2 m diameter ducts would be used to provide auxiliary ventilation in other development headings.

Mine Air Heating

Heating of the intake air would be required during the winter months to prevent water freezing underground and to provide acceptable conditions for underground workers and equipment. Mine air would be heated to +2 C by a direct-fired propane heater located at the portal. A parallel portal and drift would house the propane heater infrastructure. The air in the propane heater drift would be heated and blended with the cold air entering through the main decline, allowing vehicles and personnel to enter the mine without the use of double air lock doors. The similar air heating arrangement would be constructed at the collar of intake ventilation raise.

It was estimated that approximately 3.1 M litres of propane per year would be required to heat the mine air during five months from approximately November to April.

Underground Electrical Power Distribution System

The major electrical power consumption in the mine would be from the following:

- Main and auxiliary ventilation fans (surface and UG);
- Drilling equipment (raiseborers, jumbo, rockbolter and exploration drill);
- Freezing plant (surface);
- Underground backfill plant (crusher and pump);
- Mine dewatering pumps;
- Air compressors; and
- Maintenance shop (surface).

A high voltage cable would enter the mine via the decline and be distributed to electrical sub-stations located on mine sub-levels. The power cables would be suspended from the back of development headings. All equipment and cables would be fully protected to prevent electrical hazards to personnel.

High voltage power would be delivered at 4.16 kV and reduced to 600 V at electrical sub-stations. All power would be three-phase. Lighting and convenience receptacles would be single-phase 120 V power.

The following list of equipment would require power usage for surface and underground equipment.

Table 15.7: Mine Electrical Power System Requirements

Description	Quantity	Unit (kW)	Load Factor (%)	Utilization (%)	Power Consumption (kW/yr)
Surface					
Maintenance Shop	1	150	80	20	210,200
Surface Misc. (office, lighting, etc.)	1	65	80	40	182,200
Main Ventilation Fan	1	200	85	100	1,489,200
Underground					
Jumbo, two-boom	1	135	95	60	674,100
Rockbolter	1	135	95	60	674,100
Exploration Drill	1	75	95	50	312,100
Raiseborer	3	300	95	50	3,744,900
Backfill Cone Crusher	1	110	95	50	457,700
Backfill Pump	1	15	95	50	62,400
Portable Compressor	2	100	80	30	420,500
Feeder	2	10	95	50	83,200
Conveyor	1	15	95	50	62,400
Portable Welder	1	34	80	10	23,800
Auxiliary Fan, 75 kW	2	75	80	90	946,100
Auxiliary Fan, 50 kW	2	50	80	90	630,700
Refuge Chamber	3	5	80	100	105,100
Main Dewatering Pump	1	100	85	67	498,900
Portable Pump	2	15	85	50	111,700
Sub-total					10,689,300
Miscellaneous Power Allowance	10%				1,068,900
Freezing Plant East Zone	1	665	95	100	5,518,800
Freezing Plant West Zone	1	1455	95	100	12,106,600
Total Power East Zone					**17,277,000**
Total Power West Zone					**23,865,000**

The expected mine power draw would be approximately 17.3 MW for the East zone. It would increase to 23.9 MW when mining would start at the West zone

Underground Communication System

A leaky feeder communication system would be used as the primary communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the electrical sub-stations, refuge stations, backfill plant, and main sump.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) would be supplied with an underground radio for contact with the leaky feeder network.

Compressed Air

The mobile drilling equipment such as jumbos, rockbolters, and scissor lifts with ammonium nitrate/fuel oil (“ANFO”) loaders would be equipped with their own compressors. No reticulated compressed air system was envisioned to be required.

Two portable compressors would be required to satisfy compressed air consumption for miscellaneous underground operations, such as: jackleg and stoper drilling, secondary pumping with pneumatic pumps.

Explosives Storage and Handling

Explosives would be stored on surface in permanent magazines. Detonation supplies (NONEL, electrical caps, detonating cords, etc.) would be stored in a separate magazine. Day boxes would be used as temporary storage underground for daily explosive consumption.

Ammonium nitrate (“AN”) and fuel oil (“FO”) would be used as the major explosive for mine development. Packaged emulsion would be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques may be used as required in main access development headings, with the use of trim powder for loading the perimeter holes.

During the decline development, blasting in the development headings would be done at any time during the shift when the face is loaded and ready for blast. All personnel underground would be required to be in a designated Safe Work Area during blasting. During production period, a central blast system would be used to initiate blasts for all loaded development headings at the end of the shift. All blasting in the mine would be development-style blasting. No large scale blasts (“mass blasts”) would be undertaken.

Fuel Storage and Distribution

Haulage trucks, LHDs, and all auxiliary vehicles would be fuelled at fuel stations on surface. The fuel/lube cassette will be used for refuelling and lubrication of face equipment such as jumbo and rock bolter.

An average fuel consumption rate of approximately 3,360 l/d is estimated for the period of full production as shown in Table 15.8.

Table 15.8: Underground Mining Fuel Consumption

Description	Quantity	Consumption (l/hr)	Load Factor (%)	Utilization (%)	Total Fuel (l/day)
LHD, 5.4 m3	2	68	75	50	850
Truck, 30 t	2	80	75	50	1,000
Jumbo, two-boom	1	22	75	20	55
Rockbolter	1	22	75	20	55
Grader	1	36	75	30	135
ANFO Loader	1	22	75	30	80
Cassette Carrier	2	27	75	50	340
Mechanics truck	1	22	75	50	140
Scissor Lift	2	27	75	50	340
Supervisor Vehicle	2	18	75	30	165
Electrician Vehicle	1	22	75	50	140
Forklift	1	16	75	30	60
Total					**3,360**

Mine Dewatering

The main sources of water inflow to the underground mine are anticipated to be from groundwater and drilling operations. A detailed hydrogeological study would be required to estimate underground water inflow rates but these are expected to be minimized with ground freezing and grouting plans.

Old remuck bays, planned to be located every 150 m along the access decline, would be utilized as temporary sumps during main access decline development.

The main sump would typically be a two-bay design to allow suspended solids to settle out of the water before pumping. It would be located at the bottom level of the mine.

Water was planned to be pumped from the main sump by a high-pressure pump through a 6" diameter steel pipe located in the intake ventilation raise to the final tailing pump box on surface. The sump would be equipped with two high-head submersible pumps – one for operation and one on standby.

It is assumed that ground water would be contaminated with radon gas. The mine sump design should consider the direction of ventilation air flow. The dewatering pumps would be located on fresh air and the exhaust should directly go to the exhaust raise.

Transportation of Personnel and Materials Underground

All mine supplies and personnel would access the underground workings via the main access decline.

Two personnel carriers would be used to shuttle men from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel-powered Toyota trucks as transportation underground. Mechanics and electricians would use the mechanics' truck and maintenance service vehicles.

A boom truck with a 10t crane would be used to move supplies, drill parts, and other consumables from surface to active underground workings.

Underground Construction and Mine Maintenance

A mine service crew will perform the following:

- Mine maintenance and construction work;
- Ground support control and scaling;
- Road checking and maintenance;
- Construction of ventilation doors, bulkheads, and concrete work;
- Mine dewatering; and
- Safety work.

An underground grader and scissor lift will be utilized to maintain the main declines and active work areas.

Equipment Maintenance

Mobile underground equipment would be maintained in a mechanical shop located on the surface. Some small maintenance and emergency repairs would be performed underground. A mechanics truck would be used to perform emergency repairs underground.

Major rebuild work would be conducted off site.

A maintenance supervisor would provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. He would also provide training for the maintenance workforce.

A maintenance planner would schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators would provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

Mine Safety

The portable refuge stations would be provided in the main underground work areas; on the drilling and mucking levels and at the bottom of the mine. The refuge stations would be designed to be equipped with compressed air (medical grade), potable water, and first aid equipment; they would also be supplied with a fixed telephone line and emergency lighting. The refuge chambers can be sealed to prevent the entry of gases.

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and wherever a fire hazard exists. Every vehicle would carry at least one fire extinguisher of adequate size and proper type. All underground mobile vehicles would be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at the main access decline portal and the collar of intake raise to alert underground workers in the event of an emergency.

The main access decline would provide primary access and the intake ventilation raise with dedicated manway would provide the secondary exit in case of emergency.

Radon Monitoring

Continuous radon progeny monitoring devices would be used in the all mining working places. Local lamp-boxes would be installed to alert personnel of action levels to be taken. All regulatory measures as related to radioactive dusts would be adhered to in the mine operations planning and training

15.5.6 Mine Equipment

Criteria used in the selection of underground mining equipment include:

- Mining method;
- Material handling method;
- Mineral deposit geometry and dimensions; and
- Mine production rate;
- Ventilation requirements;
- Operating and capital cost.

Table 15.9 lists underground mobile equipment required for the mine production.

Table 15.9: Underground Major Mobile Equipment List

Equipment	Type	Quantity
Drilling Equipment		
Development Jumbo	2 boom	1
Rockbolter		1
Raiseborer		4
Exploration Drill		1
Loading & Hauling Equipment		
Development LHD	5.4 m^3	1
Production LHD	5.4 m^3	1
Haulage Truck, development	30 t	1
Haulage Truck, production	30 t	1
Service Vehicles		
Grader		1
ANFO Loader		1
Cassette Carrier		2
Personnel Cassette		2
Boom Cassette		1
Fuel / Lube Cassette		1
Mechanics Truck		1
Scissor Lift		1
Supervisor/Engineering Vehicle		2
Electrician Vehicle - Scissor Lift		1
Shotcrete Sprayer		1
Transmixer		1
Forklift		1

The equipment list was developed based on the scheduled quantities of work and estimated from first principle cycle times and productivities (83% operational efficiency was used accounting for 50 minutes of usable time in one operating hour). Some other efficiency factors such as: 80% efficiency for the second boom on the drill jumbo, 90% fill factors for LHD and trucks, additional time for remote control loading, travel, setup and teardown were used in cycle time estimations. The number of operating units was calculated based on 83% shift efficiency (shift change, lunch break, and equipment inspection time were excluded from the shift hours) and then converted to a fleet size by accounting for 80% equipment mechanical availability.

Stationary equipment was selected and would be installed and used for the following:

- Primary and auxiliary ventilation;
- Ground freezing;
- Material handling;
- Backfill;
- Compressed air;
- Mine water management;
- Underground electrical;
- Communication;
- Mine safety;
- Explosives storage; and
- Engineering equipment.

15.5.7 Personnel

The mining employees at the Roughrider underground operation were divided into two categories: salaried personnel and hourly labour.

The personnel requirement estimates were based on the crew rotation of two 12-h shifts per day with two crews working on site and two crews off site.

A mining contractor would begin work in the pre-production development stage of the mine life to allow time for the Owner to recruit staff for the project. The labour and personnel requirements described in this section do not include pre-production development, which would be performed by the contractor.

Salaried personnel requirements, including engineering, technical, and supervisory staff, are listed in Table 15.10.

Table 15.10: Technical and Supervisory Staff

Manpower Description	Quantity
Staff Mine Operation	
Mine Superintendent	1
Mine Captain	1
Mine Supervisor/Shift Boss	4
Chief Mining Engineer	1
Mine Ventilation/Project Engineer	1
Chief Geologist	1
Production Geologist	1
Geologist Technician/Sampler	4
Chief Surveyor	1
Surveyor/Mine Technical	1
Total Operating Staff	**16**
Staff Mine Maintenance	
Maintenance Superintendent	1
Maintenance Planner	1
Mechanical/Electrical Foreman	1
Raiseboring Foreman	1
Maintenance Supervisor/Shift Boss	2
Total Mine Maintenance Staff	**6**
Total Mining Staff	**22**

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance and services requirements.

Hourly labour requirements at full production are listed in Table 15.11.

Table 15.11: Hourly Personnel

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
HOURLY MINE LABOUR			
Production / Development			
Jumbo Operator	1	2	4
Ground Support	2	4	8
Raiseborer Operator	3	6	12
Raiseborer Helper	3	6	12
Haulage			
Scoop-Loader Operator	2	4	8
Truck Drivers	1	2	4
Exploration			
Diamond Driller	1	1	2
Diamond Drill Helper	1	1	2
Mine Services & Safety			
Backfill Plant Worker	2	2	4
Construction/ General Labourer	2	2	4
Grader Operator	1	1	2
Utility Vehicle Operator/Nipper	1	1	2
General Helper	1	1	2
Sub-total Mine Operating	**21**	**33**	**66**
MINE MAINTENANCE			
HD Mechanic, mobile	1	2	4
Mechanic, stationary	1	2	4
Mechanic, raiseboring	1	1	2
Electrician	1	2	4
Welder	1	1	2
Tireman/Instrument Man	1	1	2
Dry/Lampman/Bitman	1	2	4
Sub-total Mine Maintenance	**7**	**11**	**22**
Total Mine Operating	**28**	**44**	**88**

15.5.8 Mine Development Schedule

The mine development is divided into two periods: pre-production development (prior to mine production) and ongoing development (during production).

The objective of the pre-production development would be to provide access to the mineralized zones and establish all required mine services to support mine production, such as: ventilation, freezing, material handling, backfill, water management.

Pre-production development was scheduled to:

- Provide access for trackless equipment;
- Provide ventilation and emergency egress;
- Establish material handling systems;
- Install mining services (power distribution, communications, explosives storage, water supply, freezing services, mine dewatering); and
- Provide optimum development in advance of start-up to develop sufficient mineral resources to support the mine production rate.

It was assumed that all underground pre-production development would be performed by a contractor. Approximately one month would be required for mobilization of mining equipment and crews to the site and establish the required services. The jumbo crew would develop a portal and start developing decline.

The development schedule was developed based on estimated cycle times for jumbo development and best practises of North American contractors. It was assumed that the advance rate of the main decline development from the portal to the elevation about 20 m below unconformity would be approximately 60 m/month and takes into account the grouting cycle that will be required to minimize water ingress. The advance rate of decline development in good basement ground below unconformity would be approximately 150 m/month per single heading. When multiple headings are available, the advance rate per jumbo crew could be increased to 250 m per month.

The ventilation raise development was planned to be done by a raiseboring crew. It was estimated that a raiseboring crew would drill approximately 750 m per month of pilot holes. Reaming to the 4.0 m diameter would be done at an advance rate of 300 m per month and 4.5 m diameter at 250 m per month.

It is estimated that the pre-production development would be completed in four years. The development in the pre-production period was assumed as capital development. During mine production, the development for entire mineralized zones such as: freezing, drilling and mucking levels as well as ventilation drifts and raises, were considered as capital development, but development of drilling and mucking drifts inside the mineralized zones was included in operating costs.

Table 15.12: Underground Development Schedule

Development	Unit	Y -4	Y -3	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Total
Jumbo Crew Mobilization	LS	1														1
Main Access Decline	m	540	720	1,011	315											2,586
East Zone Freeze Drift	m			368												368
East Zone Freeze Holes	m				5,000											5,000
East Zone Drilling Level	m				645											345
East Zone Mucking Level	m				761											461
Haulage Level	m				228											228
Ventilation Drift	m				151											151
Crosscuts	m			110	107				250	58						525
Remuck Bays	m	45	75	90	15											225
West Zone Upper Freeze Drift	m								202							202
West Zone Lower Freeze Drift	m								326							326
West Zone Freeze Holes	m									15,743						15,743
West Zone Mucking Level	m								500	450						950
West Zone Drilling Level	m										458					458
Production Development	m					443	443	443	443	443	516	592	592	592	260	5,099
Raise Crew Mobilization	LS			1												1
Raise Freeze Holes	m			4,000												4,000
Intake Raise	m			230	125											355
Exhaust Raise	m				355											355
West Internal Raise	m										110					110
Central Internal Raise	m								62							62
High Grade Ore Pass	m				17				64							81
Low Grade Ore Pass	m				17				64							81
Main Sump	LS				1											1
Underground Backfill Chamber	LS			1												1
Total Jumbo Development	**m**	**585**	**795**	**1,579**	**2,222**	**443**	**443**	**443**	**1,721**	**959**	**974**	**592**	**592**	**592**	**260**	**12,200**
Total Raise Development	**m**			**230**	**514**				**190**		**110**					**1,044**

15.5.9 Mine Production Schedule

The underground mine production at the Roughrider deposit was scheduled to support mill feed at a required daily capacity of 200 t of mineralized material at a grade of 3.3% U_3O_8.

The raiseboring productivities were estimated from first principles for a raise size of 3.05 m diameter. Productivity of one raiseboring crew could be up to 133 t/day. Assuming that multiple production faces would be available all the time, three raiseboring crews would fully satisfy the production schedule.

It was assumed that required mill feed grade of 3.3% U_3O_8 could be achieved with appropriate scheduling and sequencing of raiseboring production and proper blending of mineralized material.

The Roughrider production schedule is based on the estimated Mining Inventory, which included 286,000 t Indicated Resources at 2.5% U_3O_8, and 446,000 t Inferred Resources at 3.8% U_3O_8. The Roughrider mine life was estimated to be about 14.5 years including four years of pre-production period.

The mine production by year is shown in Table 15.13.

Table 15.13: Underground Production Schedule

Parameter	Unit	Year											Total
		1	2	3	4	5	6	7	8	9	10	11	
Mill Feed Material													
East Zone	Kt	70.4	70.4	70.4	70.4	70.4	34.5						387.9
West Zone	Kt						35.9	70.4	70.4	70.4	70.4	28.1	344.2
Total Mill Feed	Kt	70.4	70.4	70.4	70.4	70.4	70.4	70.4	70.4	70.4	70.4	28.1	732.1
Mill Feed Grade	U_3O_8%	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3	3.3
Contained U_3O_8	U_3O_8 Klb	5,122	5,122	5,122	5,122	5,122	5,122	5,122	5,122	5,122	5,122	2,044	53,262
Production Waste													
East Zone	Kt	17.7	17.7	17.7	17.7	17.7	9						
West Zone	Kt						16.8	34.3	34.3	34.3	34.3	13.7	167.6
Total Production Waste	Kt	17.7	17.7	17.7	17.7	17.7	25.8	34.3	34.3	34.3	34.3	13.7	265.2
Total Production Material	Kt	88.1	88.1	88.1	88.1	88.1	96.2	104.7	104.7	104.7	104.7	41.8	997.3

16 Process Description

16.1 Process Selection

The selected process for the Roughrider PEA is based on the results of the test work completed in the first two phases of test work as described above. The milling process used for the preparation of mill capital and operating cost estimates for Hathor's Roughrider uranium project was a grind/acid leach/resin-in-pulp process. This process was used as a basis of the scoping study estimates but remains to be further assessed with further metallurgical testing and trade-off studies as mining plans are further developed and as the project advances. Changes to unit operations and control strategies will occur in conjunction with metallurgical design developments. Consequently, this process description will require revisions from time to time to reflect these changes.

The run-of-mine ore will be received at the mill site and fed to a primary crusher as a blended feed. Grinding will be accomplished in a SAG (Semi-Autogenous Grinding) mill/ball mill grinding circuit. The ground slurry will be thickened and leached in a series of mechanically agitated leach tanks under atmospheric pressure using sulphuric acid and sodium chlorate oxidant to maintain oxidizing conditions.

Uranium recovery from leach liquor will be accomplished using ion exchange in a resin-in-pulp circuit. Resin ion exchange has been selected for the PEA in place of solvent ion exchange (extraction) in part for the following reasons:

- The achievable thickened leach residue densities [approximately 35% solids (w/w)) are low relative to the requirements of a typical counter-current-decantation ("CCD") circuit (low underflow densities imply that numerous CCD stages would be required to for effective uranium recovery from the leach slurry);
- Capital costs can be reduced by up to 15 %;
- Equal or lower operating costs compared to a CCD/SX circuit;
- No requirement to keep large volumes of organic on site; and
- No impact from residual organics in treated effluent.

Uranium precipitation will be accomplished using hydrogen peroxide with magnesia for pH control. The uranium product will be packaged in 205 L drums for shipment to the uranium refinery.

Tailings and effluent treatment will be accomplished with the treatment conditions currently in use at Saskatchewan uranium mines. This will include reverse osmosis and multiple chemical treatment stages with discharge of neutralized tailings to the tailings management facility and treated effluent to the environment. Permeate from reverse osmosis will be recycled to the mill for use as a process water.

A summary flow diagram of the process is depicted in the flowsheet (No. 533-F-100) shown below as Figure 16.1.

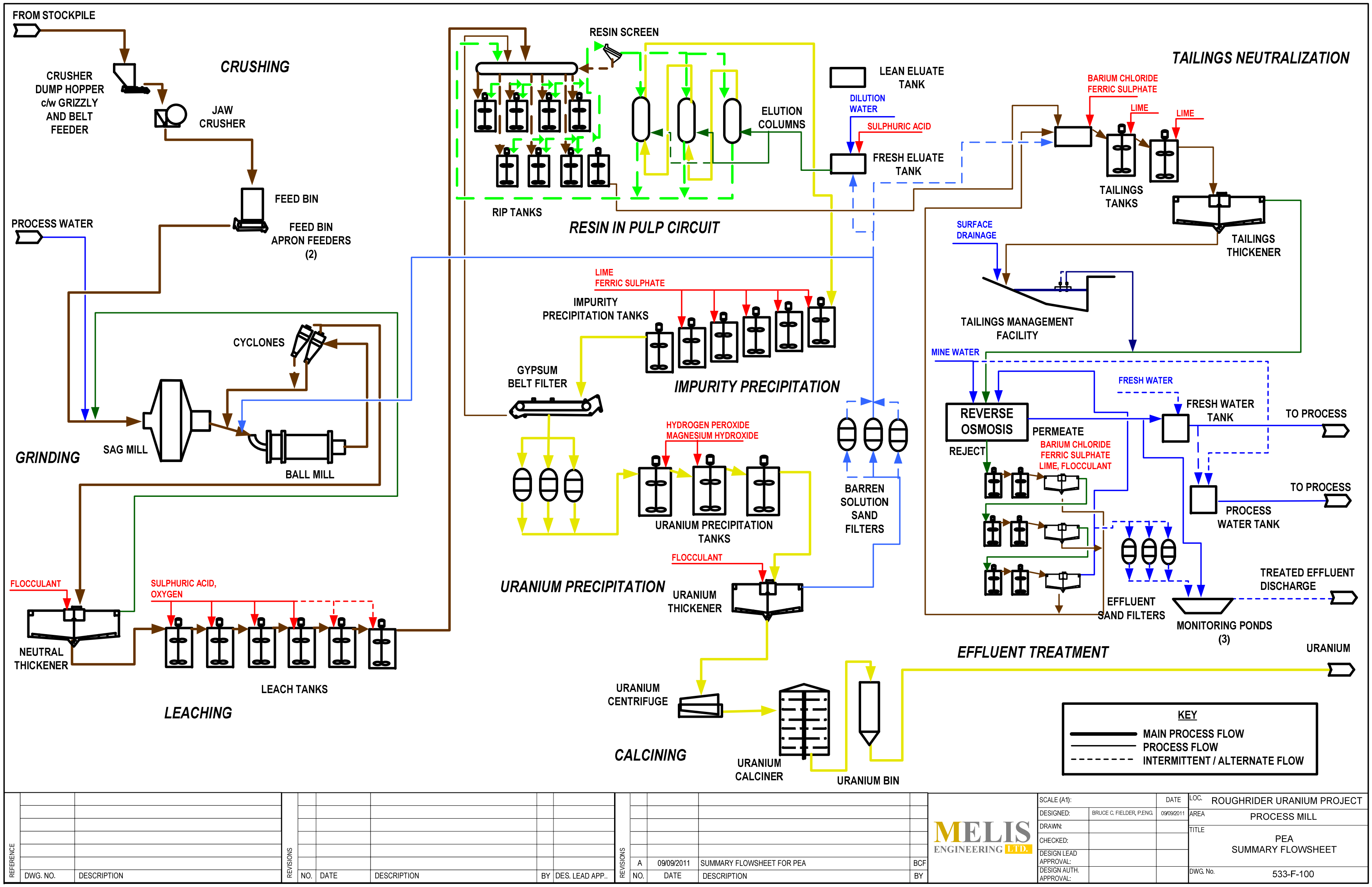


Figure 16.1: Summary Flow Diagram

16.2 Production Basis

The main assumptions used for the mill capital and operating cost estimates are as follows:

- Annual Production – 5,000,000 lbs U_3O_8 (2,267,962 kg U_3O_8);
- Mill Feed Grade – 3.3% U_3O_8;
- Net Uranium Recovery – 97.7%;
- Mill Throughput;
  - Average 70,322 tonnes/year;
  - Design – 78,137 tonnes/year (based on 90% mill availability and 350 days per year operation);
  - Instantaneous
    - 223 tonnes/day;
    - 9.30 tonnes/hour.
- Processing Parameters – from Phases I and II metallurgical test work on Roughrider West Zone;
- Process – Grind/Leach/Resin-In-Pulp (“RIP”).

16.3 Process Overview

Run-of-mine ore, at an average grade of 3.30% U_3O_8, will be received as high grade and low grade mineralization and stockpiled close to the mill for feeding a blend into a dump hopper and jaw crusher. Crushed ore will be discharged to a coarse ore bin to provide some surge capacity ahead of a SAG mill and ball mill grinding circuit. Ground slurry, at a product K_{80} size (80% passing size) of 125 microns, will be thickened prior to feeding the leach circuit.

Leaching will be completed under atmospheric conditions in mechanically agitated rubber lined steel tanks using sulphuric acid and sodium chlorate oxidant. A free acid level of 15 g H_2SO_4/L, an oxidation/reduction potential of 475 mV maintained with approximately 4 kg $NaClO_3$/t, a 50 °C leach temperature and 12 hours retention time will provide a leach uranium extraction efficiency of 98.5%. Accounting for soluble and insoluble losses the overall uranium recovery is estimated at 97.7%.

The soluble uranium in the leach slurry will be recovered in an eight stage resin-in-pulp ion exchange circuit with uranium selective resin advancing counter-currently to the leach slurry flow. The loaded resin will be eluted with sulphuric acid to produce a concentrated liquor of the recovered uranium.

Resin technologies for uranium recovery have advanced sufficiently in recent years to make this type of circuit a viable option to a CCD (counter current decantation)/solvent extraction circuit, and overall capital costs have been shown to be lower.

The concentrated eluate from the resin-in-pulp circuit will be partially neutralized for gypsum removal ahead of uranium (yellowcake) precipitation with hydrogen peroxide and magnesium oxide for pH control. The yellowcake product will be calcined or dried prior to packaging in standard 205 L drums for shipment to a uranium refinery.

Leach residue and waste solution streams will be bulk neutralized with lime and treated with barium chloride and ferric sulphate prior to discharge to the tailings management facility. Tailings decant water and excess mine water will be treated in a four stage treatment circuit, a first stage reverse osmosis (“RO”) circuit followed by three stage treatment of the RO reject using lime, barium chloride and ferric sulphate. RO permeate will be recycled to the mill for use as a process water. The treated effluent will pass through sand filters to remove residual suspended solids prior to discharge to the environment.

16.4 Reagents

Table 16.1 below summarizes reagents used in the process plant circuits:

Table 16.1: Summary of Process Reagent Usage

Reagent	Used in Process Plant Circuit
Barium Chloride	Tailings Treatment Effluent Treatment
Ferric Sulphate	Impurity Precipitation Tailings Treatment Effluent Treatment
Flocculant	Neutral Thickener Product Thickener Tailings Thickener Effluent Treatment Clarifiers
Hydrogen Peroxide	Uranium Precipitation
Lime	Impurity Precipitation Effluent Treatment Tailings Neutralization
Magnesium Oxide	Uranium Precipitation
Resin	Resin-In-Pulp
Sodium Chlorate	Leaching
Steel Balls	SAG Mill, Ball and Lime Slaking Mill
Sulphuric Acid	Leaching Resin elution Effluent Treatment
Yellowcake Drums	Packaging

16.5 Utilities

Water supply to the process plant consists of fresh water, process water and minewater. Reverse osmosis permeate is the main source of supply for fresh water, which is used wherever pure water is required. Process and mine water are used wherever pure water is not required.

Steam is used to heat the leach slurry to 50ºC.

Process and instrument air are supplied to the process plant. Two process air compressors supply process air; a bleed stream of process air is dried and de-oiled for use as instrument air.

A safety shower system is installed in the process plant. The system consists of a head tank containing warmed potable water and at least one safety shower/eye wash combination in each process circuit.

17 Project Infrastructure

Access to the project site is proposed to be by a 7 km long all-weather gravel road that connects to Provincial Highway 905 to the project. See Figure 17.1. As planned, the road would pass over one small wetland area that would require a bridge or culvert.

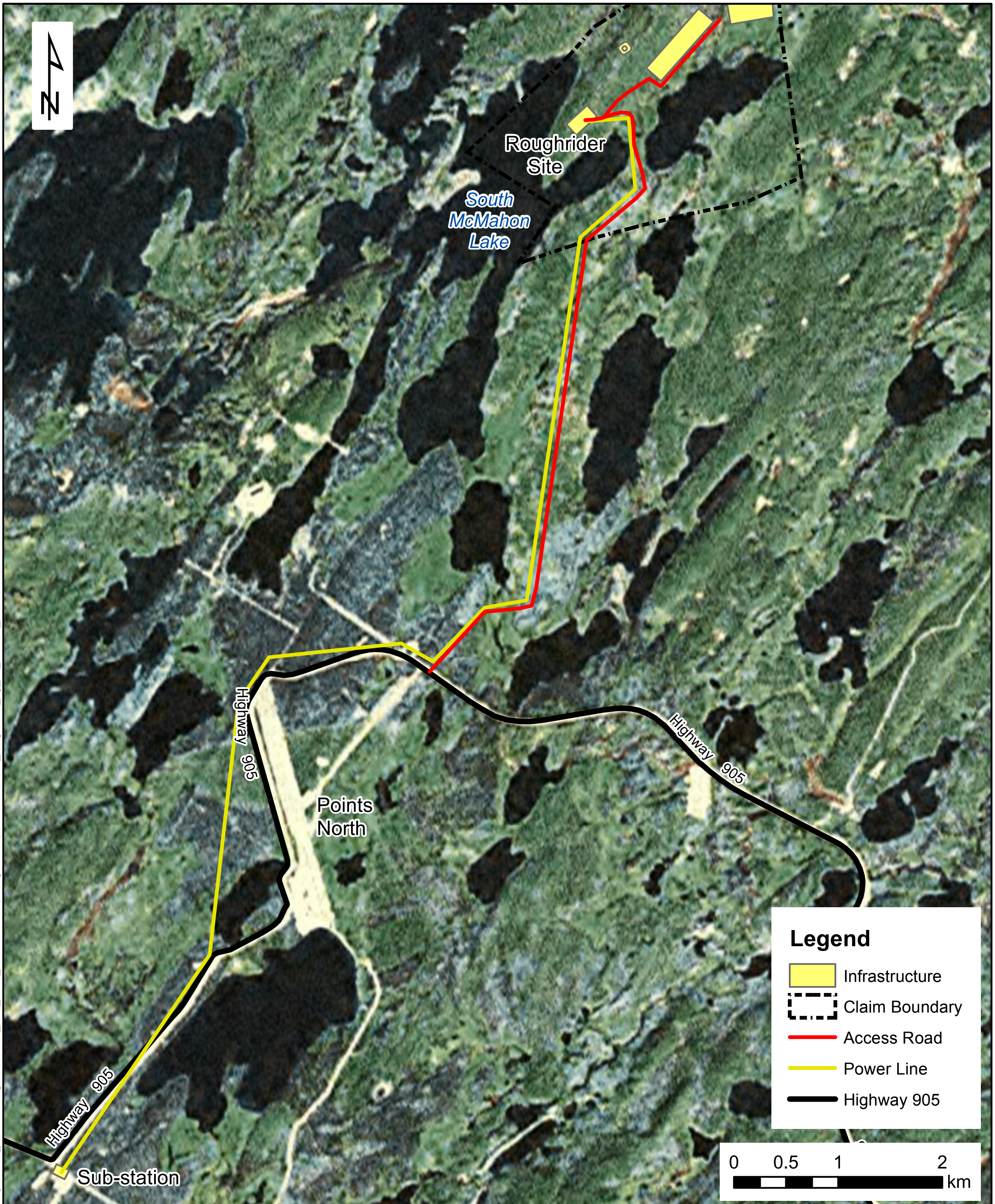
Power to the site is envisioned to come from the existing SaskPower substation southwest of Points North. The power transmission line to the site would be approximately 13 km long.

Camp facilities would be built on the property to house the on-going operation and an additional temporary camp would be utilized during construction. Facilities at Points North would be used for back-up services during short periods of high camp demand.

A 1.5 MW surface freeze plant would be built to support all UG freezing operations.

Process water would be sourced from recycled water and additional make-up water coming from the mine, wells or from a nearby lake.

A detailed surface layout was not developed for this study but a general, approximate plan is shown in Figure 17.2.



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HATHOR EXPLORATION LIMITED

Preliminary Economic Assessment Study

Roughider Access Plan

Job No: 2CH012.000

Filename: 2CH012.000_roughrider_fig_01_access_plan

Roughrider Project

Date: October 2011

Approved:

Figure: 1

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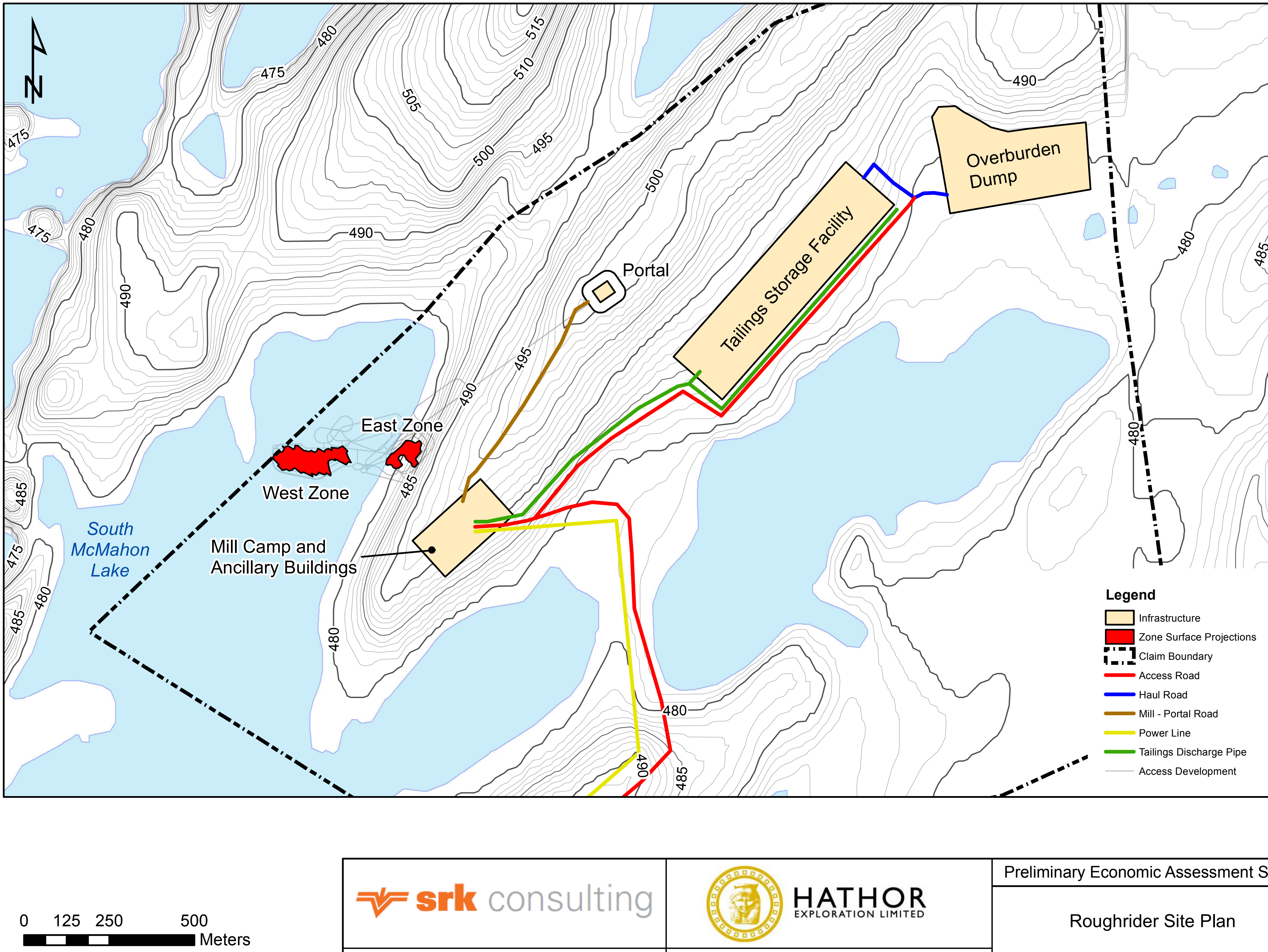

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HATHOR EXPLORATION LIMITED

Preliminary Economic Assessment Study

Roughrider Site Plan

Job No: 2CH012.000

Filename: 2CH012.000_roughrider_fig_02_site_plan

Roughrider Project

Date: October 2011

Approved: -

Figure: 2

17.1 Waste Management

17.1.1 Tailings

Introduction

Tailings will be stored in a purpose-built excavation situated on a gently sloping side hill approximately 1 km northeast of the proposed plant site. The principal design objectives for the tailings storage facility ("TSF") require that all tailings be kept below the local water table and that there be a positive gradient into the TSF from the surrounding natural ground. The latter objective will be met by using the pervious surround method, which is based on the placement of permeable, granular material, plus a filter layer, beneath and around the entire TSF perimeter. Pumps will be installed in the granular material to enable the water level in this material to be maintained below the local groundwater table.

At closure, the tailings will be capped with a filter layer, granular material, and a layer suitable for the establishment of a vegetative cover.

The current concepts associated with the TSF design, site selection, construction, operation and closure reflect industry best practices. Further details are provided below.

Design Criteria

The design of the TSF commenced with the establishment of the key design criteria. For the PEA, these were taken to consist of the following:

- A single storage facility should be developed to store all tailings produced over the life of mine.
- The required storage volume should allow for ice entrainment of at least 15% over the operating life of the TSF.
- In consideration of the preceding two criteria and a final settled tailings dry density of 0.9 tonnes per cubic meter, the required storage volume was taken as 1,000,000 m^3.
- The stored tailings must be kept below the local water table and a positive gradient into the TSF from the surrounding natural ground must exist at all times.
- In view of the abundance of lakes in the vicinity of the property, and in order to allow for appropriate groundwater monitoring and, if needed, groundwater controls, a setback of 100 m to the nearest water body was arbitrarily established.

These criteria were used as the basis for site selection and design.

Site Selection

Topographic data was used to identify a series of potential TSF sites within a radius of approximately 3 km of the proposed plant site. These sites were compared on the basis of factors such as distance to the plant site, elevation relative to the plant site, conditions along the likely pipeline route and the site specific terrain. Based on these site comparison parameters, a site situated on a gently sloping side hill approximately 1 km northeast of the proposed plant site was selected (Figure 17.2).

It is noteworthy that the comparison revealed that the differences between the identified sites were relatively minor. If, for some reason, other parameters need to be factored into the TSF site selection at the next stage of the project, there appears to be other potential sites in relatively close proximity to the proposed plant site.

Conceptual Design

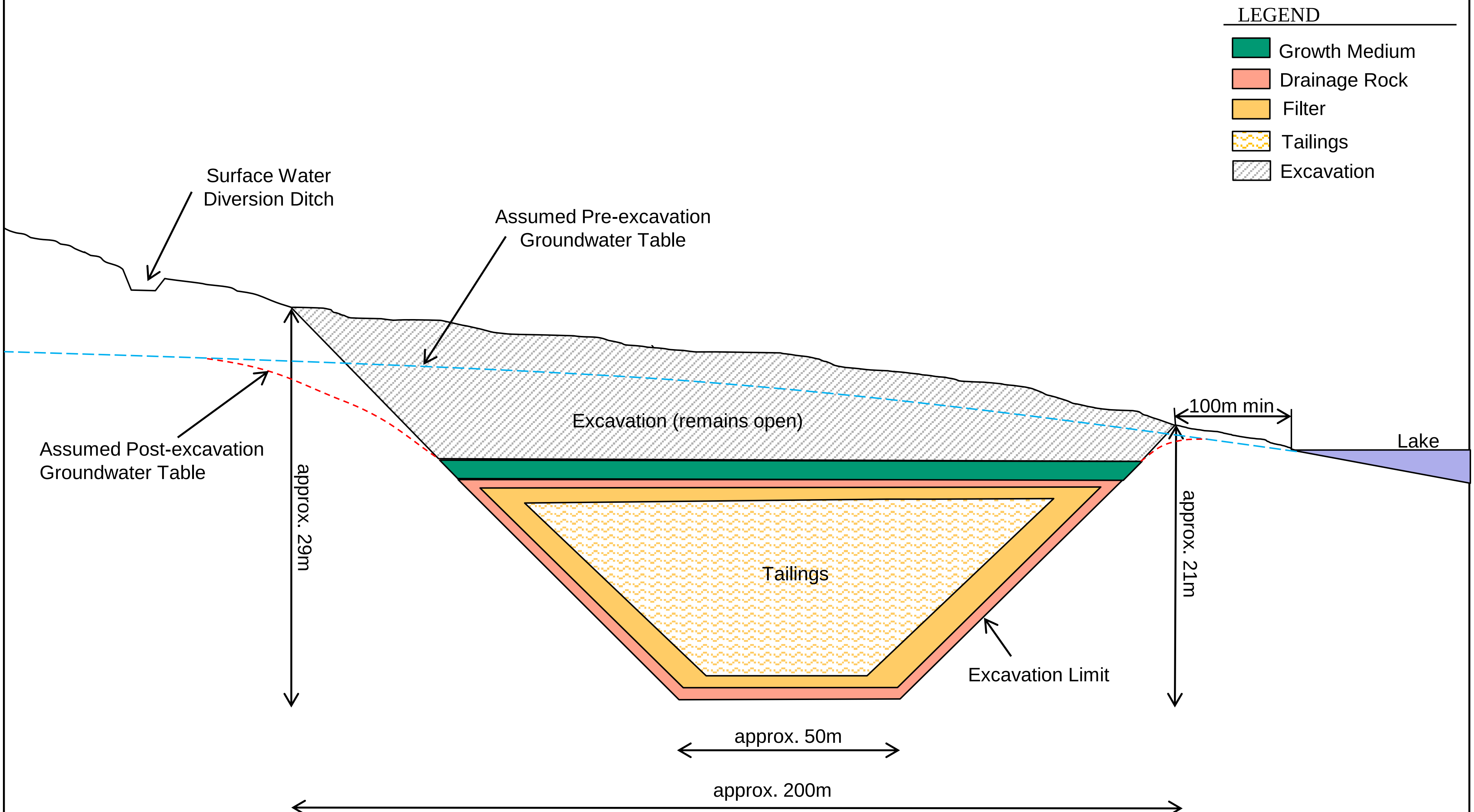
The conceptual design of the TSF is driven by the requirement that all tailings will be kept below the local water table and that there will be a positive gradient into the TSF from the surrounding natural ground. Therefore, the design will consist of a purpose-built excavation which will be lined with permeable, granular material, plus a filter layer, beneath and around the entire TSF perimeter.

The positive gradient will be achieved by installing a pump in the granular material to keep the water level in this material below the local groundwater table.

A typical section through the proposed TMF is shown in. The excavation will be about 200 m wide and 750 m long, and will extend Figure 17.3 to a maximum depth of about 30 m. No site specific geotechnical data is available at the TSF site, but regional data suggests the site coincides with a drumlin. In this case, the local soil is likely to consist mainly of a dense basal till. The design assumes the side slopes of the excavation will be excavated at 2.5H:1V (horizontal:vertical).

In the absence of site specific hydrogeological data, an assumed water table was selected (Figure 17.3), with a minimum elevation at the nearby lake level (477 masl). Based on this assumption, at least the bottom 17 m or so of this excavation will be below the existing local water table. It is within this portion of the excavation that the tailings will be stored.

Prior to tailings deposition, a granular drainage layer will be placed on the bottom and around the sides of the excavation. In addition, to prevent potential clogging of the drainage layer with tailings, a filter layer consisting of crushed rock or washed sand with specified particle size distribution will be placed over/against the drainage layer. Relative layer thicknesses are illustrated on Figure 17.3.



Notes:
- Not to scale
- Groundwater to remain above tailings surface

srk consulting

HATHOR EXPLORATION LIMITED

Preliminary Economic Assessment Study

Conceptual Tailings Storage Facility – Cross Section

Job No: 2CH12.000	Roughrider	Date: Oct. 25,, 2011	Approved: CCS
Filename: Figure xxxx.pptx		Figure: **17.3**	

The horizontal thickness of the filter layer will be in the order of 4 m, while the vertical thickness on the bottom and top will be around 1 m. The drainage layer will be constructed to a thickness of about 1 m all around.

Construction

The entire pit will be excavated as part of initial construction. The excavated material will be transported and placed in an overburden dump immediately to the northeast of the TSF.

Current concepts call for the placement of the drainage and filter layers within the TSF in accordance with the following three stages:

- Stage I - construction of the base drainage layer and filter, followed by raising the side drainage and filter zones by 7 m, to a sufficient height to allow tailings deposition for about 4 years.
- Stage II – raising the permeable surround by 4 m to allow tailings deposition for about 3 years.
- Stage III – raising the permeable surround to its full height of 15 m.

The proposed staging will spread the construction costs over time and will also limit the duration of time that the drainage/filter layers are exposed to potential damage before being covered by tailings.

Operation

The procedures associated with the operation of uranium tailings impoundments in northern Saskatchewan are fairly well established, and it is expected that these general procedures will be followed at this TSF.

The tailings will be transported hydraulically in a pipeline from the plant site and discharged into the TSF. Excess water will be recycled to the plant site.

A pump within the perimeter drain will be used to ensure that negative head gradients are maintained within the TSF during the active deposition phase. Water pumped from the drainage layer will be re-used in the mill or treated and disposed of, as appropriate.

Closure

At the end of the mine life, the TSF will be closed by first placing a multi-layer cap over the tailings. From bottom to top, the cap will consist of a 1 m layer of filter material, a 1 m layer of drainage rock and a 2 m layer of soil obtained from the overburden dump that will be suitable as a growth medium. The surface will be shaped and/or ditched to direct surface water off the TSF surface.

The thickness and elevations of the various layers will be re-evaluated at closure in order to be sure the entire tailings volume and all or part of the drainage layer is located below the natural water table. The surface of the growth medium will be situated above the water table to prevent groundwater discharge at the surface of the closed TSF.

Pumping will then be stopped and the natural water table will be allowed to re-establish.

17.1.2 Waste Rock

Details regarding the waste rock have not been addressed as part of this PEA, primarily because it is anticipated that most or all of the waste rock will likely be used as underground backfill prior to mine closure.

In the interim period prior to closure, waste rock that reports to surface will be stored on a pad or pads. The details of the pad design will depend on the geochemical characteristics of the waste rock see Section 19.4.2 for further details on the geochemistry of the waste rock). It is anticipated that all clean waste rock will be stockpiled on an unlined pad. Geochemically problematic waste rock will likely require a more secure storage, possibly involving a liner and water management system.

The location and detailed design of the pad(s) will be engineered in the next phase of project engineering.

18 Market Studies and Contracts

18.1 Market Studies

SRK did not perform a market study to try and predict the long-term uranium price. The uranium prices selected for the three economic cases in the PEA were:

- Case A (US$60/lb U_3O_8): Approximate current long-term contract prices
- Case B (US$70/lb U_3O_8): Estimate of potential future long-term contract prices
- Case C (US$90/lb U_3O_8): Optimistic price scenario

18.2 Contracts

SRK is not aware of any significant existing contracts that might affect the project.

19 Environmental Studies, Permitting and Social or Community Impact

In Saskatchewan the environmental assessment and permitting framework for the development of a mining project consists of a two tiered system. The first tier consists of an environmental assessment (EA) phase involving departments from both the federal and provincial governments. Following a successful EA the project would proceed to the second tier of regulation which consists of a construction and operating licensing/permitting phase again involving both federal and provincial government departments and agencies. The project is then regulated through all phases (construction, operation, closure and post closure) by the same federal and provincial departments and agencies.

Unique to uranium, which is classified as a strategic mineral under federal legislation, the Canadian Nuclear Safety Commission (CNSC) a federally established commission reporting to the federal cabinet through the Minister of Natural Resources Canada was established in 2000 to regulate *the use of nuclear energy and materials to protect the health, safety and security of Canadians and the environment; and to implement Canada's international commitments on the peaceful use of nuclear energy* (CNSC, 2011).

Uranium has been mined in Saskatchewan since the mid-1900s. The development of new deposits in the late 1970s (Cluff Lake uranium mine) saw an increase in public interest/concern with uranium mining in the province. This public interest/concern has been present with the onset of each new uranium development in the Province since Cluff Lake mine. As a result government (federal and provincial) and industry have increased their attention to addressing social considerations associated with uranium mining in Saskatchewan.

19.1 Environmental Assessment

19.1.1 Provincial Requirements

In the Province of Saskatchewan the Environmental Assessment Act is administered by the Ministry of Environment (MOE). The level of assessment for mining projects is dependent on the specific characteristics of each individual project. The Ministry follows the following process to determine which level of assessment will be required.

In Saskatchewan, the proponent of a project that is considered to be a “development” pursuant to Section 2(d) of the Environmental Assessment Act is required to conduct an environmental impact assessment (EIA) of the proposed project and prepare and submit an environmental impact statement (EIS) to the Minister of Environment.

Section 2(d) of the Environmental Assessment Act reads:

“development” means any project, operation or activity or any alteration or expansion of any project, operation or activity which is likely to:

- Have an effect on any unique, rare or endangered feature of the environment;
- Substantially utilize any provincial resource and in so doing pre-empt the use, or potential use, of that resource for any other purpose;
- Cause the emission of any pollutants or create by-products, residual or waste products which require handling and disposal in a manner that is not regulated by any other Act or regulation;
- Cause widespread public concern because of potential environmental changes;
- Involve a new technology that is concerned with resource utilization and that may induce significant environmental change; or
- Have a significant impact on the environment or necessitate a further development which is likely to have a significant impact on the environment; (Sask. Env. Act, 2002).

The Roughrider Project as it is currently defined meets the Province’s definition of a “development” and will therefore be required to conduct a Provincial EIA.

19.1.2 Federal Requirements

A federal environmental assessment is required for a proposed project if it meets the definition of a “project” as defined in the Canadian Environmental Assessment Act (CEAA) and a federal authority has certain decision making responsibilities. The CEAA is triggered when a federal authority:

- Is the proponent
- Provides financial assistance to a proponent in order to advance the project
- Has an ownership position in the land or grants an interest in the land allowing the project to proceed
- Has a regulatory responsibility for a component of the project (CEAA, 2007)

In accordance with Canada's Nuclear Safety and Control Act every uranium development in Canada requires regulatory decisions by the Canadian Nuclear Safety Commission (CNSC) and therefore triggers CEAA and a federal assessment in all cases.

Once a federal assessment is triggered the agency then determines what level of an EA the project will require. The CEAA defines four possible levels of assessment, Screening Level, Comprehensive Study, assessment by Mediator or a Review Panel. It is anticipated, based on the current scope of the Roughrider project, that a federal Comprehensive Study assessment, at a minimum, will be required.

In addition to the federally legislated requirements defining the need for an environmental assessment the federal government introduced the Major Projects Management Office (MPMO) in 2007. The MPMO role is to provide a management and coordinating role for major resource development projects in Canada. The authority and mandate of the office is provided through a Committee comprised of Deputy Ministers from federal departments typically identified as "responsible authorities" in the conduct of a federal environmental assessment. The MPMO has no legislative authority. The MPMO typically screens the project descriptions of proposed projects and then issues a determination as to whether or not their office should play a administrative role in the execution of a federal EA.

Saskatchewan and Canada honour a cooperation agreement which harmonizes the two assessment processes to run concurrently under a single administrative process. This process is typically administered jointly by Saskatchewan's Assessment Branch (Regina, Saskatchewan) and the CEAA regional office located in Winnipeg, Manitoba.

19.2 Licensing and Permitting

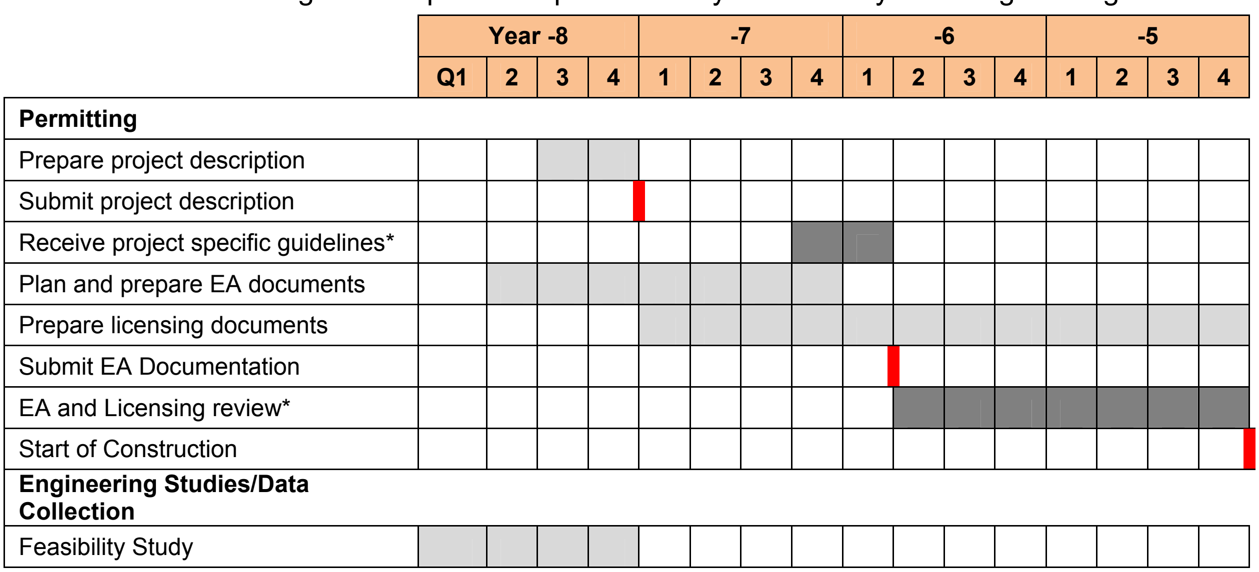
In the event the environmental assessments are deemed complete and Ministerial approvals are granted by the respective provincial and federal Ministers, the project will be allowed to proceed to the second tier of environmental regulation. This requires the proponent to obtain a variety of approvals and permits again from both levels of government.

Federally, separate approvals to construct and operate are required from the CNSC. Depending on the specific components of the project addition approvals may also be required by the department of Fisheries and Oceans Canada ("DFO") and Environment Canada ("EC"). The federal licensing process requires the submission of detailed engineering design packages as well as detailed Management Plans for all facets of the operation as part of their licensing process.

Provincially, approvals to construct are required from the Ministry of Environment. Following all necessary approvals to construct the “pollutant control facilities” as defined in the Mineral Industry Environmental Protection regulations (MIEPB) an “Approval to Operate” will be developed for the project which outlines all provincial environmental monitoring and reporting requirements.

19.3 Assessment Schedule and Estimated Costs

Based on the scope of the project it is expected that it will be required to undergo a Comprehensive Study Assessment in conjunction with a provincial assessment. Using previous assessments of similar projects for comparison along with advise from regulatory agencies it is estimated that the environmental assessment of the Roughrider project will require a minimum of 36months from submission of the Project Description to receipt of environmental assessment approvals to proceed with the project as shown in Figure 19.1. It should be noted that this timeline likely represents the best case timing. Actual timing may vary from 36 to 48 months. The variation in schedule will be a function of the complexity of the proposed project and how it interacts with the environment. Therefore the accuracy of the schedule can only be refined following the completion of pre-feasibility or feasibility level engineering studies.



**Actual duration of these tasks are dependent upon Federal and Provincial regulatory processes.*

Figure 19.1: Conceptual EA and Licensing Schedule

Costs associated with completing an Environmental Assessment are also a function of the complexity of the project, the level of Assessment the project must undergo and the commodity involved. When uranium is the commodity the Canadian Nuclear Safety Commission (CNSC) is the lead federal agency involved in the Assessment.

This Agency operates on a cost recovery basis which allows the agency to bill the proponent for each hour their staff dedicates to the Assessment process which complicates the ability to accurately estimate the total costs of an assessment.

A reasonable estimate of the total costs associated with completing an Environmental Assessment for this project, at this stage of its development, is approximately $ 5 M.

19.4 Environmental Considerations

The most significant environmental considerations associated with the Roughrider project are centered around the management of the waste streams associated with an underground uranium mine and mill project. With the primary focus being placed on the potential impacts associated with the management of tailings, mill effluent and mine water and the proponent’s commitment to effectively mitigate the potential impacts associated with these waste streams, throughout operations, decommissioning and reclamation and post reclamation.

Engineering and environmental investigations required for final characterization of the potential impacts and to subsequently support the final design of the waste management facilities have not yet been completed.

Based on the existing data there were no environmental fatal flaws identified with this proposed project.

19.4.1 Tailings

The current level of engineering associated with the tailings storage facility reflects industry best practices. As such, it is reasonable to assume the design proposed in this study will not be considered fatal flaws during the proposed project’s environmental assessment provided that the appropriate levels of engineering and environmental investigations are completed as part of the project’s next level of engineering. For a more detailed description of the tailings management facility refer to section 17.4.1 of this report.

19.4.2 Waste Rock

The development of this project has the potential to generate three types of waste rock, clean waste rock, potentially acid generating and/or metal leaching (PAG/ML) waste rock and/or special waste rock.

Follow up engineering studies will require the completion of a detailed waste rock geochemical characterization program and subsequently the development of a waste rock management plan.

This plan will segregate the waste rock into two categories, clean waste rock and PAG/ML and/or special waste. The designation of special waste is defined as waste containing 0.03 % U_3O_8 to 0.10 % U_3O_8. In the event special waste and/or PAG/ML waste is identified this material will require storage on a lined pad and a specific management plan to handle this material. All seepage and runoff collected on this pad will require treatment prior to discharge to the environment.

All clean waste rock will be stockpiled on an unlined pad. The location and detailed design of this pad will be engineered in the next phase of project engineering.

19.4.3 Mine Water

All mine water collected in the underground will likely require treatment to remove radionuclides and other potential contaminants. This water may be routed back to the mill to be used as process water. If there is excess water then this water will be treated prior to being released to the environment.

Careful consideration to the discharge location of the treated effluent is recommended. Discharge to larger water bodies is typically preferred by the regulatory and scientific communities while recent public consultations with residents of northern Saskatchewan suggest their preference would be to discharge treated effluents into smaller water bodies which are not as heavily utilized for traditional, commercial and recreational pursuits.

19.5 Social Considerations

Significant efforts have been expended by the Saskatchewan government and the uranium mining industry since the early 1990s to solicit and incorporate Traditional Knowledge, concerns and desires of northern Saskatchewan Residents (both aboriginal and non-aboriginal) into the environmental assessment process. There are a number of well-established forums and committees in existence, in northern Saskatchewan, mandated to facilitate consultation between the proponents of proposed uranium developments and stakeholder groups as well as provincial and federal regulatory requirements for both industry and governments to do the same.

The main stakeholder groups that Hathor's Roughrider project will be required to interact with throughout all phases of the project life (Environmental Assessment, operations, closure and post closure) will be:

- Hatchet Lake First Nations;
- Fond du lac First Nation;
- Black Lake First Nation;
- Community of Stony Rapids;
- Community of Uranium City
- Community of Camsel Portage;
- Prince Albert Grande Council; and
- Northern Saskatchewan Environmental Quality Committee.

Previous assessments involving the above stakeholder groups have shown the fundamental areas of concern involve the development and implementation of robust environmental management plans throughout operations, coupled with a closure plan that ensures very low risk of long term environmental impacts.

The main stakeholder groups involved with this project's assessment will be looking for assurance that culturally sensitive lands, lakes and traditionally harvested foods (ungulates, fish and berries) will not be significantly impacted as a result of the development. From a socio-economic perspective, many if not all of these communities and political entities have interests in limited partnerships and other business ventures established to take advantage of the economic opportunities associated with northern Saskatchewan's mining industry. These stakeholder groups will be looking for opportunities to enter into contractual arrangements to maximize the involvement of these businesses with the project in the event the project gains environmental assessment approvals to proceed.

Representatives of Hathor have initiated a stakeholder consultation process that represents a free, prior and informed consultation process which is consistent with international standards for consultations with indigenous peoples.

19.6 Conceptual Decommissioning and Reclamation Plan

Based on the current scope of the project a robust, while economically realistic, closure plan can be developed for the Roughrider Project. The goals of this closure plan would be to:

- Restrict or eliminate the migration of all potential contaminants of concern from all sources on the mine site;
- Restrict or eliminate all potential radiological exposures to animal or humans;
- Restrict or eliminate all potential public safety risks associated with the decommissioned and reclaimed mine site; and
- Return the property, to the extent possible, to pre-mining conditions with the ultimate goal of returning the property to Provincial Institutional Control.

To meet the above goals all mine infrastructure would be removed. All clean waste rock piles would be contoured to a stable slope angle. The piles and disturbed areas would be scarified, vegetated and if necessary, re-vegetated and fertilized on a regular frequency until such time that the vegetation was self-sustaining. All PAG/ML waste would be covered with an appropriate cover designed to limit infiltration and encourage runoff. The underground mine would be allowed to flood. No water would be allowed to discharge to the environment from the Provincial surface lease without meeting the site specific closure requirements which would be developed in consultations with the regulatory, community and stakeholder groups prior to implementation of the decommissioning and reclamation plan.

Depending on the results of the geochemical characterization of waste rock it is possible that there are limited volumes of PAG/ML associated with this project. In which case it is possible the PAG/ML rock could be utilized as backfill in the underground workings, which would be a viable closure option for this material. The water treatment plant would remain operational until such time that all mine water sources were below or met the site specific decommissioning and reclamation criteria.

20 Capital and Operation Costs

20.1 Operating Costs

20.1.1 Summary

Unit OPEX costs were developed from first principles with inputs based on budgetary consumable prices, contractor feedback, experience and cost estimation services. Table 20.1 shows a summary of the LOM unit OPEX estimate.

Table 20.1 Unit OPEX Estimate

Operating Costs	Unit ($)	Unit OPEX Estimate
Mining	$/t milled	421
Processing	$/t milled	480
General and Administration	$/t milled	126
Average Unit operating Cost	**$/t milled**	**1,026**
Average Unit operating Cost	**$/lb U_3O_8**	**14.4**

20.1.2 Mine

All costs are expressed in Canadian dollars, with no allowance for escalation or interest during construction. The estimate was prepared at a ±35%scoping study level of accuracy. No contingency was applied to cost estimate but instead, was applied in the economic model.

Underground Mining Operating Cost Estimate

The underground mining operating cost was estimated for the owner operating scenario. All costs are estimated in Canadian dollars per tonne of mill feed material at the rate of 200 t/d.

The underground mining operating costs were calculated for each cost category such as: development, raiseboring production, ground freezing maintenance, material handling, backfill, mine services, maintenance, exploration, and safety. Labour cost was estimated based on manpower list and was included in total operating cost as separate cost category.

The costs were estimated using a combination of first principles calculations, experience and factored costs.

Table 20.2 shows the input data for cost estimation that were assumed in this study.

Table 20.2: Operating Cost Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/a	352
Nominal Mining Rate	t/d	200
Average Mining Rate	t/a	70,400
Rock Characteristics		
In Situ Density of Mineralization	t/m^3	2.59
In Situ Density Waste	t/m^3	2.55
Swell Factor	%	55
Loose Density of Mineralization	t/m^3	1.67
Loose Density Waste	t/m^3	1.65
Shift Data		
Working Days per Week	ea.	7
Shifts per Day	ea.	2
Shift Length	hr	12
Shift Change	hr	1
Lunch Break	hr	0.75
Equip Inspection	hr	0.25
Subtotal Non-productive	hr	2
Usable Time per Shift	hr	10
Shift Efficiency	%	83
Usable Minutes per Hour	min	50
Hour Efficiency (50 min/h)	%	83
Effective Work Time per Shift	hr	8.3

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation. Supply costs were based on estimated supply consumption and recent Canadian supplier’s prices. Maintenance consumables, such as parts, tyres, etc., as well as fuel ($1.20 /L), propane ($0.80 /L), and power ($0.06 /kWh) were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs.

Stoping operating costs were estimated for the West and East zones separately as each zone would require different amount of development, raiseboring, backfilling, and freezing.

The operation development cost was based on amount of required production development for drilling and mucking levels for each zone, assuming that development would be performed by owner jumbo crew. Total cost of production development was divided to the tonnage of mineralized material per each zone to estimate operating cost per tonne.

The raiseboring production cost was estimated based on raiseborer productivities, amount of drilling and reaming required through the waste and mineralized material, raiseboring consumables and equipment requirements. The raiseboring cost for the West zone ($71.65 /t) is higher than raiseboring cost for the East zone ($60.30 /t) as per higher amount of drilling through the waste material in the West zone.

The cost of ground freezing included maintaining the ground freezing around ventilation raises and production areas. The freezing cost of $6.83 per tonne for the East zone and $12.45 per tonne for the West zone was estimated based on power required to maintain freezing ($7*10^{-5}$ MW per meter freeze pipe) and an allowance for freeze plant and piping maintenance.

The material handling cost of $6.66 per tonne was estimated based on mucking and trucking costs of waste and mineralized material.

Backfill cost of $32.20 per tonne for the East zone and $38.52 per tonne for the West zone was based on amount of backfill requirements for production raises and backfill of production development. Cement cost of $340 per tonne was used in backfill cost estimation.

The mine services cost of $65.06 per tonne was estimated based on equipment working time, materials, propane and power supply required for ventilation, air heating, compressed air, transportation of personnel and materials, mine and road maintenance, mine dewatering, and underground construction.

The mine maintenance cost of $10.45 per tonne was estimated based on required maintenance equipment, tools and supplies for maintenance shop. Maintenance consumables, such as parts, tyres, etc., were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs. The maintenance labour costs were included in the overall underground mine labour costs.

Mine safety and training costs of $1.89 per tonne were estimated based on the number of underground mine personnel, required personal protective equipment, first-aid and safety supplies, mine rescue, and safety training.

The underground labour cost of $206.63 per tonne was estimated based on operating ($119.50 /t) and maintenance ($42.60 /t) hourly labour requirements and salaried ($44.52 /t) personnel. The labour rates used for determining the overall mining cost were based on experience for similar operations in North America. The labour costs include base salary, production bonuses, and 36.8% of salary burden to cover leave pay, pension and superannuation benefits, insurance coverage and educational assistance.

Table 20.3: Average Salaried Personnel Cost

Staff Description	Quantity	Base Salary ($/year)	Loaded Salary ($/year)	Total Cost ($/year)
Mine Superintendent	1	140,000	205,520	205,520
Mine Captain	1	125,000	183,500	183,500
Mine Supervisor/Shift Boss	4	95,000	139,460	557,840
Chief Mining Engineer	1	120,000	176,160	176,160
Mine Ventilation/Project Engineer	1	95,000	139,460	139,460
Chief Geologist	1	110,000	161,480	161,480
Production Geologist	1	90,000	132,120	132,120
Geological Technician/Sampler	4	75,000	110,100	440,400
Chief Surveyor	1	100,000	146,800	146,800
Surveyor / Mine Technician	1	80,000	117,440	117,440
Sub-total Mine Operating Staff	16			2,260,720
Maintenance Superintendent	1	125,000	183,500	183,500
Mechanical/Electrical G. Foreman	1	105,000	154,140	154,140
Raiseboring G. Foreman	1	105,000	154,140	154,140
Maintenance Planner	1	80,000	117,440	117,440
Maintenance Supervisor/Shift Boss	2	90,000	132,120	264,240
Sub-total Mine Maintenance Staff	6			873,460
Total Mining Staff	**22**			**3,134,180**

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 12 h/d with 4 crews in rotation: two on-site and two off.

Table 20.4: Average Hourly Labour Cost

Staff Description	Quantity	Base Salary ($/hr)	Loaded Salary ($/year)	Total Cost ($/year)
Jumbo Operator	4	42	144,225	576,899
Ground Support/Services	8	38	130,489	1,043,912
Raiseborer Operator	12	42	144,225	1,730,696
Raiseborer Helper	12	38	130,489	1,565,868
LHD Operator	8	38	130,489	1,043,912
Truck Driver	4	35	120,187	480,749
Diamond Driller	2	35	120,187	240,374
Diamond Drill Helper	2	30	103,018	206,035
Backfill Plant Worker	4	35	120,187	480,749
Utility Vehicle Operator/Nipper	2	30	103,018	206,035
Construction/General Labourer	4	32	78,613	439,542
Grader Operator/Road Maintenance	2	30	103,018	206,035
General Helper	2	28	96,150	192,300
Sub-Total Mine Operating Labour	66			8,413,104
HD Mechanic, mobile	4	45	154,526	618,106
Mechanic, stationary	4	45	154,526	618,106
Mechanic, raiseboring	2	45	154,526	309,053
Electrician	4	45	154,526	618,106
Tireman/Instrument Man	2	35	112,522	225,044
Welder	2	35	112,522	225,044
Dry/Lampman/Bitman	4	30	96,448	385,790
Sub-Total Mine Maintenance Labour	22			2,999,249
Total Mine Labour	**88**			**11,412,353**

Summary of total underground mining operating cost is shown in Table 20.5.

Table 20.5: Underground Mining Operating Cost Estimate Summary

Cost Distribution	East Zone ($/t)	West Zone ($/t)
Production Development	15.16	20.25
Raiseboring	60.3	71.65
Freeze Maintenance	6.83	12.45
Material Handling	6.66	6.66
Backfill	32.2	38.52
Sub-total Stoping Cost	**121.16**	**149.53**
Services	65.06	65.06
Maintenance	10.45	10.45
Exploration	2.27	2.27
Mine Safety, Training, Mine Rescue	1.89	1.89
Sub-total Mine Services Cost	**79.67**	**79.67**
Labour Operating	119.5	119.5
Labour Maintenance	42.6	42.6
Salaried Personnel	44.52	44.52
Sub-total Mine Labour	**206.63**	**206.63**
Total Mining Operating Cost	**407.46**	**435.83**

The mining operating cost was estimated as an average over the mine life for each mineralized zones.

20.1.3 Processing Facility

Summary

Site services personnel and mill operating costs have been estimated by Melis as an input to the Roughrider PEA's total estimated operating costs for a stand-alone uranium mining/milling operation.

The estimated site services personnel and mill operating costs for a Grind/Leach/Resin-In-Pulp process are estimated at $480/tonne or $6.70/ lb U_3O_8, including a 15% allowance for unforeseen costs, as summarized in Table 20.6.

The estimated mill operating costs include all operating and maintenance costs including all mill personnel, reagents and consumables and power (based on grid power), environment, health and safety personnel and warehouse personnel. Mining costs, general site costs, camp and employee turnaround costs and other support costs such as administration, human resources, head office and overall management are not included.

Table 20.6: Site Services Personnel and Mill Operating Cost Estimates

Item	\$/year	\$/year	\$/lb U_3O_8
Personnel (Mill Operation and Maintenance)	11,292,800	160.59	2.26
Personnel (Warehouse, HSE, and Site Services)	2,241,000	31.87	0.45
Reagents and Consumables	14,009,863	199.22	2.80
Comminution Media	328,800	4.68	0.07
Electrical Power (at \$0.06/kWh)	899,000	12.78	0.18
Unforeseen Costs (15%)	4,316,000	61.37	0.86
Total	33,087,463	471	6.62
Rounded Value		480	6.70

Basis of Mill Operating Cost Estimate

The main assumptions used for the mill operating cost estimates, based on a one week in-one week out operating schedule, are the same as those used for the mill capital cost estimate, namely:

- Annual Production – 5,000,000 lbs U_3O_8 (2,267,962 kg U_3O_8);
- Mill Feed Grade – 3.3% U_3O_8;
- Mill Throughput;
  - Average 70,322 tonnes/year
  - Design – 78,137 tonnes/year (based on 90% mill availability and 350 days per year operation);
  - Instantaneous;
    - 223 tonnes/day
    - 9.30 tonnes/hour.
- Net Uranium Recovery – 97.7%;
- Processing Parameters – from Phase I and II metallurgical testwork on Roughrider West Zone;
- Process – Grind/Leach/Resin-In-Pulp (RIP);

The battery limits of the mill operating cost estimates are receipt of run-of-mine ore at the mill feed dump hopper to drum packaging of yellowcake, discharge of tailings to the tailings containment area and discharge of treated mill effluent to the environment.

Personnel Cost Estimate

The mill and site services personnel are based on the labour loading for a one week in-one week out schedule for a 24 hours per day, seven days per week operating schedule. The personnel cost are summarized in Table 20.7 below.

Table 20.7: Mill and Site Services Personnel Cost Estimates

Position	No.	Salary ($/year)	Burden (35%)	Total ($/year)
Mill Superintendent	1	150,000	52,500	202,500
Chief Metallurgist	1	130,000	45,500	175,500
Mill General Foreman	1	130,000	45,500	175,500
Metallurgists	2	100,000	35,000	270,000
Mill Foremen	4	110,000	38,500	594,000
Mill Operators	40	75,000	26,250	4,050,000
Mill Helpers	16	60,000	21,000	1,296,000
Assayers	2	90,000	31,500	243,000
Assay Technicians	4	70,000	24,500	378,000
Metallurgical Technicians	4	70,000	24,500	378,000
Sub-Total Mill Operation	75			7,762,500
Mill Maintenance				
Maintenance Superintendent	1	125,000	43,750	168,750
Mill Maintenance Planners	2	75,000	26,250	202,500
Mill Maintenance Foremen	2	110,000	38,500	297,000
Electrician/Instrument Technologists	4	100,000	35,000	540,000
Millwrights	12	100,000	35,000	1,620,000
Apprentices	8	65,000	22,750	702,000
Sub-Total Mill Maintenance	29			3,530,250
Total Mill Personnel Cost Estimates	104			11,292,800
SH&E, Warehouse, Site Services				
Overall Site Supervisor	1	120,000	42,000	162000
Clerks, etc.	3	70,000	24,500	283500
SH&E Supervisor	1	130,000	45,500	175500
SH&E Technicians	4	70,000	24,500	378000
Warehouse Supervisor	2	90,000	31,500	243000
Warehouse Technicians	4	70,000	24,500	378000
Site Services Supervisor	2	110,000	38,500	297000
Site Services Workers	4	60,000	21,000	324000
Total Site Services Personnel Cost Estimate	21			2,241,000

Reagents and Consumables Cost Estimate

The cost of process related reagents and consumables are summarized in Table 20.8 below.

Table 20.8: Estimated Costs of Reagents and Consumables

Reagents and Consumables	$/kg FOB Minesite	t/year	$/year
Barium Chloride ($BaCl_2.2H_2O$)	0.85	32	27,200
Sodium Hydroxide (NaOH)	0.60	100	60,000
Ferric Sulphate (12% Fe^{3+})	0.45	410	184,500
Flocculant A	3.90	10	39,000
Flocculant B	4.65	3	13,020
Fuel (Heating)			200,000
Diesel (Mobile Equipment)	1.50	100	150,000
Diesel (Steam Generation)	1.50	110	165,000
Diesel (Yellowcake)	1.50	900	1,350,000
Hydrogen Peroxide (70% H_2O_2)	1.00	638	638,000
Lime (98% CaO)	0.35	13,900	4,865,000
Magnesia (MgO)	0.80	235	188,000
Resin Replacement	6,500/m^3	114	742,000
Sodium Chlorate ($NaClO_3$)	1.00	281	281,288
Sulphur	0.225	12,400	2,790,000
Laboratory Supplies			250,000
Maintenance Supplies			1,080,000
Product Drum Replacement			120,000
Product Drum Liners			22,700
Product and Drum Freight			844,155
Total Reagents and Consumables Estimated Costs			14,009,863

Comminution Media Cost Estimate

The cost of comminution (grinding) media is summarized in Table 20.9 below.

Table 20.9: Estimated Costs of Comminution Media.

Steel	Consumption kg/t		FOB Minesite		$/year
	kg/t	sets per year	$/kg	$/set	
Grinding Balls - SAG	0.5	-	1.60	-	56,400
Grinding Balls - Ball Mill	0.5	-	1.60	-	56,400
SAG Mill Liners	-	1.0	-	125,000	125,000
Ball Mill Rubber Liners - Lifters		0.67		35,000	23,500
Ball Mill Rubber Liners - Shell	-	0.33	-	50,000	16,500
Jaw Crusher Jaw Plate - Stationary	-	6	-	6,000	36,000
Jaw Crusher Jaw Plate - Movable		3		5,000	15,000
Total Comminution Media Estimated Costs					328,800

Power Cost Estimate

Power cost estimates, summarized in Table 20.10 are based on estimated power requirements by unit operation using a grid power charge of $0.06/kWh as provided by Hathor.

Table 20.10: Mill Power Cost Estimates

Area	Total Estimated Operating Power kW	Total Estimated Operating Power Cost ($/year)
Crushing	91	43,700
Grinding	198	95,000
Leaching	40	19,200
Resin and Uranium Circuits	289	138,700
Tailings and Water Treatment	214	102,700
Acid Plant	450	216,000
Reagents, Water and Air	149	71,500
Buildings	442	212,200
Total Power Cost Estimates	1,873	899,000

20.1.4 General and Administration

General and administration costs were estimated as per Table 20.11.

Table 20.11: G&A Costs

Staff Administration	Quantity	Total ($/year)	Unit cost ($/t)
Mine Manager	1	$245,700	$3.49
Secretary / Receptionist	1	$70,200	$1.00
Human Resources Manager	1	$108,000	$1.53
Environmental Engineer	1	$121,500	$1.73
Warehouseman	2	$148,500	$2.11
Purchasing Agent	1	$121,500	$1.73
Safety and Training Coordinator	1	$108,000	$1.53
Chief Accountant	1	$121,500	$1.73
Accounting Clerk	1	$74,250	$1.05
Payroll Clerk	1	$74,250	$1.05
IT Services	1	$81,000	$1.15
Nurse/First Aid	4	$270,000	$3.84
Janitors/Carpenters	1	$67,500	$0.96
Surface Services and Maintenance	2	$135,000	$1.92
Safety Training & Mine Rescue		$100,000	$1.42
Medical Services and Supplies		$100,000	$1.42
Communications		$100,000	$1.42
Office Supplies		$50,000	$0.71
Computer Hardware and Software		$50,000	$0.71
Recruitment and Training		$100,000	$1.42
Consultants		$200,000	$2.84
External Assays/Testings		$100,000	$1.42
Environmental Monitoring Expenses		$150,000	$2.13
Site Operation and Maintenance Supplies		$200,000	$2.84
Site Road & Facilities Maintenance		$200,000	$2.84
Powerline Maintenance		$100,000	$1.42
Crew Transportation		$1,320,800	$18.76
Accomodation/Camp Costs		$3,013,075	$42.80
Insurances		$230,000	$3.27
Legal and Audit Fees		$100,000	$1.42
Permitting, Regulatory Compliance		$53,000	$0.75
Contingency	15%	$925,031	$13.14
TOTAL		$8,838,806	$125.55

20.2 Capital Costs

20.2.1 Summary

The estimated capital cost of the project is $567 M as shown in Table 20.12.

Table 20.12: Capital Cost Estimate Summary

Description	Unit	Estimate
UG Mine Development & Equipment	M$	159
Process Plant	M$	172
Infrastructure	M$	53
Owner's costs	M$	8
EPCM	M$	48
Closure	M$	14
Capital cost w/o contingency	M$	454
Contingency @ 25%	M$	113
TOTAL CAPITAL COST	M$	567

20.2.2 Mine

Underground Capital Cost Estimate

An underground mining capital cost is estimated at $159.1 M, including initial capital cost of $95.1 M ($30.3 M for equipment purchase and $64.8 M for underground development).

The underground mining capital cost estimate was based on the following:

- Preliminary project development plan;
- Underground mining equipment list;
- In-house database;
- Western Mining estimation references;
- Budget quotes obtained by SRK from development contractors and equipment manufacturers.

Mining capital was divided into equipment capital cost and mine development cost categories.

UG Equipment Capital Cost

The mining equipment capital cost estimate includes the purchase of permanent mining equipment required for production. The unit prices for equipment are shown in Table 20.13 were used for equipment capital cost.

Table 20.13: Underground Mining Equipment Unit Costs

Equipment	Unit	Quantity	Unit Cost (K$)
Drilling Equipment			
Jumbo (2 boom)	ea.	1	1,072
Rockbolter	ea.	1	810
Raiseborer	ea.	4	2,000
Exploration Drill	ea.	1	325
Jackleg	ea.	4	5.5
Stoper	ea.	4	4.5
Loading & Hauling Equipment			
LHD, 5.4 m^3 (10 t) with automation pac.	ea.	2	970
Haulage Truck, 30 t with lead cover	ea.	1	1,100
Haulage Truck, 30 t	ea.	1	870
Service Vehicles			
Grader	ea.	1	330
ANFO Loader	ea.	1	220
Cassette Carrier	ea.	2	275
Personnel Cassette	ea.	2	70
Boom Cassette	ea.	1	85
Flat Deck Cassette	ea.	1	70
Fuel / Lube Cassette	ea.	1	120
Mechanics Truck	ea.	1	305
Scissor Lift	ea.	1	305
Supervisor/Engineering Vehicle	ea.	2	86
Electrician Vehicle - Scissor Lift	ea.	1	115
Shotcrete Sprayer	ea.	1	700
Transmixer	ea.	1	350
Forklift	ea.	1	140
Material Handling			
Grizzly	ea.	2	40
Feeder	ea.	2	200
Truck Loading Automation System	ea.	1	400
Equipment Foundation and Installation	ea.	1	390
Backfill Plant			
Cone Crusher	ea.	1	350
Cement Storage, Screw Feeder	ea.	1	200
Mixer	ea.	1	150
Backfill Pump	ea.	1	170
Piping	ea.	1	50
Valves and Fittings	ea.	1	30

Equipment	Unit	Quantity	Unit Cost (K$)
Equipment Foundation and Installation	ea.	1	475
Freeze Plant			
Freeze Plant	ea.	1	2,500
Brine Supply and Instrumentation	ea.	1	500
Temperature Monitoring	ea.	1	200
Equipment Installation	ea.	1	150
Ventilation			
Primary Ventilation Fan	ea.	1	250
Primary Ventilation Fan Accessories	ea.	1	50
Fan Foundation and Installation	ea.	1	100
Auxiliary Ventilation Fan, 75 kW	ea.	2	30
Auxiliary Ventilation Fan, 50 kW	ea.	2	25
Direct Fired Propane Heating System	ea.	2	450
Ventilation Doors and Regulators	ea.	2	30
Compressed Air			
Compressed Air System	ea.	1	200
Portable Compressor	ea.	2	70
Mine Water Management			
Surface Water Tank, Piping and Insulation	ea.	1	20
Fresh Water Pump	ea.	2	7
Main Dewatering Pump	ea.	2	180
Portable Pump	ea.	2	10
Equipment Foundation and Installation			207
Mine Electrical			
750kVA Portable Substation	ea.	3	115
Switchgear	ea.	4	40
Cables and Distribution	ea.	1	150
Surface Transformer	ea.	1	30
Miscellaneous (15%)			
Communication System			
Leaky Feeder System	ea.	1	150
Underground Telephone	ea.	4	1
Mine Safety			
Portable Refuge Station	ea.	2	74
Gas Monitoring System	ea.	1	50
Mine Rescue Equipment	ea.	1	80
First Aid Equipment	ea.	1	20
Cap Lamps	ea.	50	0.5
Personal Protective Equipment	ea.	50	0.4
Fire Extinguishers	ea.	10	0.1

Equipment	Unit	Quantity	Unit Cost (K$)
Sanitary Unit	ea.	3	5
Sanitary Pumping Tank System	ea.	1	5
Stench Gas System	ea.	2	10
Auxiliary Men Hoist System	ea.	1	500
Foam Generator	ea.	1	15
Mine Engineering Equipment			
Survey Equipment	ea.	1	75
PC, Printers, Network, Software	ea.	1	50
Mine Design Software	ea.	1	50
Geology Department Software	ea.	1	50
Miscellaneous			
Mining Tools	ea.	1	50
Surface Repair Shop	ea.	1	230
Explosives Storage			
Underground Powder Magazines	ea.	1	60
Underground Cap Magazines	ea.	1	30

Purchasing of spare parts in amount of 5% of equipment cost was assumed at initial equipment purchase and an additional 4% of equipment cost was applied to cover expenses for delivering equipment to the site.

Underground mining equipment capital cost by year is summarized in Table 20.14.

Table 20.14: Underground Mining Equipment Capital Cost Summary

Item	Y -4	Y -3	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Total
	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)
Drilling Equipment			0.3	9.9				1.6	4.4	4						20.2
Loading and Hauling Equipment				3.9				2.9	1	2		1				10.8
Service Vehicles			0.2	3.7					1.8	1						6.7
Ventilation	0.5			1												1.5
Material Handling				0.8												0.8
Backfill				1				0.2	0.1							1.3
Compressed Air				0.3												0.3
Mine Water Management				0.4												0.4
Underground Electrical	0.2		0.3	0.3												0.8
Communication	0.2															0.2
Safety			0.1	0.8												0.9
Underground Engineering Equipment	0.2															0.2
Underground Miscellaneous				0.4												0.4
Freeze Plant		3	0.4													3.4
Spare Parts (5%)		0.2	0.1	1.1				0.2	0.4	0.3		0.1				2.4
Freight (4%)		0.1		0.9				0.2	0.3	0.3						1.8
Total	**1.1**	**3.3**	**1.4**	**24.5**				**5.1**	**8**	**7.6**		**1.1**				**52.1**

UG Development Capital Cost

All development in pre-production period is included in capital costs. During mine production, the development required for entire mineralized zones such as: freezing, drilling and mucking levels, ventilation drifts and raises, were considered as capital development, but production development of drilling and mucking drifts inside the mineralized zones was included in operating costs.

It was assumed that all development in pre-production period would be done by contractor, so the contractor rates were used accordingly for jumbo and raise development. Capital development cost included contractor mobilization/demobilization costs or both: jumbo and raiseboring crews.

Cost of ground freezing of mineralized zones was estimated based on required amount of drilling and $1,500 /m cost of drilling and pipe installation per meter of hole of hole.

Underground mine development capital costs is shown in Table 20.15.

Table 20.15: Underground Development Capital Cost

Development	Y -4	Y -3	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Total
	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)	(M$)
Jumbo Crew Mobilization	0.5										0.5
Portal Boxcut Excavation	0.5										0.5
Main Access Decline	7	9.4	10.3	2.2							28.9
East Zone Freeze Drift			2.6								2.6
East Zone Freeze Holes				7.5							7.5
East Zone Freezing				0.1							0.1
East Zone Drilling Level				3.2							3.2
East Zone Mucking Level				4.4							4.4
Haulage Level				1.4							1.4
Ventilation Drift				0.9							0.9
Crosscuts			0.6	0.6				1.7	0.3		3.2
Remuck Bays	0.2	0.4	0.4	0.1							1.1
West Zone Upper Freeze Drift								2.4			2.4
West Zone Lower Freeze Drift								2.3			2.3
West Zone Freeze Holes									23.6		23.6
West Zone Mucking Level								4	3.6		7.6
West Zone Drilling Level										3.2	3.2
Raise Crew Mobilization			0.5								0.5
Raise Collar			1								1
Raise Freeze Holes			2.4								2.4
Raise Freezing			0.1								0.1
Intake Raise			1.1	0.6							1.7
Exhaust Raise				1.8							1.8
West Internal Raise										0.3	0.3
Central Internal Raise								0.3			0.3
High Grade Ore Pass				0.1				0.2			0.3
Low Grade Ore Pass				0.1				0.2			0.3
Main Sump				0.3							0.3
Underground Backfill Chamber			0.3								0.3
Miscellaneous Development			0.2	0.2							0.4
Owner cost	1	1	1	1							4
Total	**9.2**	**10.8**	**20.5**	**24.5**				**11.1**	**27.5**	**3.5**	**107.1**

No mine development capital costs in Y7 to Y11

20.2.3 Processing Facility

Mill capital costs have been estimated by Melis as an input to the Roughrider PEA's total estimated costs for a stand-alone uranium mining/milling operation.

The estimated mill capital costs for a Grind/Acid Leach/Resin-In-Pulp process are estimated at $215,000,000, including a 25% contingency. The capital cost estimate is summarized in Table 20.16 below.

Table 20.16: Summary of Class V/ IV Mill Capital Cost Estimate

Cost Area	Cost, $ (2011)
Labour (Hours), excluding contingency	416,000
Labour Cost	49,767,000
Material and Building Cost	81,025,430
Reagents, First Fill	1,978,840
Total Direct Cost	132,771,270
Contractor Overhead and Profit (15%)	19,940,000
Engineering, Procurement, and Management (15%)	19,940,000
Total Direct and Indirect Costs	172,795,270
Contingency (25%)	43,200,000
Capital Spares, 1% of Equipment Cost	690,000
Total Estimated Capital Costs	216,685,270
Assumed:	215,000,000

Basis Of Mill Capital Cost Estimate

The main assumptions used for the mill capital cost estimates are as follows:

- Annual Production – 5,000,000 lb U_3O_8 (2,267,962 kg U_3O_8);
- Mill Feed Grade – 3.3% U_3O_8;
- Mill Throughput;
  - Average 70,322 tonnes/year
  - Design – 78,137 tonnes/year (based on 350 scheduled days per year operation and 90% mill availability)
  - Instantaneous
    - 223 tonnes/day
    - 9.3 tonnes/hour
- Net Uranium Recovery – 97.7%;
- Processing Parameters – from Phases I and II metallurgical testwork on Roughrider West Zone; and
- Process – Grind/Acid Leach/Resin-In-Pulp (RIP) as per the summary process flowsheet presented in Figure 16.1.

The battery limits of the mill capital cost estimates are receipt of run-of-mine ore at the mill feed dump hopper to drum packaging of yellowcake, discharge of tailings to the tailings management facility and discharge of treated mill effluent to the environment.

Included in the capital cost estimate are only the process related equipment and buildings, specifically:

- The process plant building complete with ore receiving and crushing;
- Analytical and metallurgical laboratories;
- Process equipment for all unit operations located in the mill building;
- Air and water utilities for the mill process;
- 125 t/day sulphuric acid plant to meet the design acid requirements;
- Reagent first fills;
- Capital spares;
- Contractor indirects;
- Engineering, procurement and construction management;
- Contingency at 25%.

The capital costs were completed on an order-of-magnitude basis to the level of detail between a Class V estimate (-20% to -50%/+30% to +100%) and a Class IV estimate (-15% to -30%/+20% to +50%). The estimate was completed in third quarter 2011 Canadian dollars.

Equipment requirements and approximate sizing were defined by the design criteria and process flowsheets, based on metallurgical data available from Phase I and Phase II testwork on Roughrider West mineralization.

The cost estimate was prepared with separate spreadsheets for each unit operation associated with each process, plus separate spreadsheets for utilities and the process building. Each spreadsheet lists the major pieces of equipment or material in that area. The installed cost of each piece of equipment or material was then estimated with the installation labour, unit cost and cost of field materials being included in the total installed mechanical cost. Line items in the capital cost estimates were costed based on budget quotes or on Melis file data.

For each unit area, the costs of process piping, electrical, instrumentation were estimated as a percentage of total mechanical equipment costs based on accepted industry standards from. The sum of these costs and the total installed mechanical cost were the total direct costs for that unit operation. Freight costs were estimated for each unit operation.

The US dollar conversion rate used was $1.05 Cdn/$ US.

The labour rate of $120/h chosen was an all-in construction hourly rate which was agreed to between Melis, SRK and Hathor.

Indirect costs included contractor overhead and profit and engineering, procurement and construction management (EPCM). Each was estimated as a percentage of the total direct costs for that option. Contractor overhead and profit was estimated at 15% of total direct costs and EPCM was estimated at 15% of total direct costs.

A 25% contingency factor was used in the estimate as agreed to between Melis, SRK and Hathor.

20.2.4 Infrastructure

Project infrastructure cost estimates are shown in Table 20.17.

Table 20.17: Infrastructure Capital Cost Estimate Summary (excluding contingency)

Description	Unit	Estimate
Access Road	M$	2.4
Power Line	M$	2.5
Tailings Management Facility	M$	30.8
Camp and facilities	M$	7.0
Buildings and other facilities	M$	10.0
Infrastructure capital cost w/o contingency	M$	52.7

Access road costs were based on local contractor estimates for a 7 km all-weather, single lane road, with pull-outs. Additional costs were allocated for the construction of through one marshy area. The remainder of the road is located on or near drumlins and, as such, the construction environment was assumed to be very favourable.

The power line to site was discussed with SaskPower and was estimate received at a cost of $2.5 M.

The cost estimate for the tailings management facility ($31 M) was done by first principles using the design described in Section 17.1.

Camp facilities and other buildings ($17 M) were allocated a cost based on experience at other sites.

21 Economic Analysis

21.1 Introduction

The economic analysis described in this report provides only a preliminary overview of the project economics based on broad, factored assumptions.

The mineral resources used in the LOM plan and economic analysis include Inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources will be upgraded to a higher resource category. Based on this, there is no certainty that the results of this preliminary assessment will be realized.

21.1.1 Assumptions

Simplified earnings before taxes (“PT”) analyses were performed based on three commodity prices; US$60/lb U_3O_8, US$70/lb U_3O_8 and US$90/lb U_3O_8 (Cases A to C respectively).

The common economic model assumptions for all cases include:

- Economic analysis starts after all studies, permitting and financing are complete. The time period for completion of studies, permitting and financing would be expected to be a minimum of 4 years;
- 7% discount rate (“DR”) for net present value (“NPV”) calculations, unless otherwise noted;
- 100% equity financing (no interest charges);
- A constant currency exchange rate of C$1.05 to US$1.00;
- Exclusion of duties and taxes;
- Inclusion of royalties under the existing royalty structure (which are currently under review);
- 100% payable U_3O_8 and no off-site costs;
- Four years of construction prior to production; and
- Constant mineable tonnes and grades derived from mineable shapes using a U_3O_8 price of US$60/lb.

21.1.2 Economic Analysis Results

Based on the Case A U_3O_8 price of US$60/lb, the pre-tax internal rate of return ("PT-IRR") was estimated to be 32% and the pre-tax net present value at a 7% discount rate ("PT-NPV$_{7\%}$") was estimated to be $769M.

The Case B economic analysis yielded an PT-IRR of 38% and an PT-NPV$_{7\%}$ of $1,025M, at a U_3O_8 price of US$70/lb.

The Case C (US$90/lb U_3O_8) economic analysis yielded a PT-IRR of 48% and a PT-NPV$_{7\%}$ of $1,527M. See Table 21.1 for a summary of the economic results.

Table 21.1: Economic Results

Parameter	Unit	Case A	Case B	Case C
U_3O_8 Price	US$/lb U_3O_8	60	70	90
Royalty Payments	M$	366	463	669
PT-NPV$_{0\%}$	M$	1,592	2,042	2,928
PT-NPV$_{7\%}$	M$	769	1,025	1,527
PT-NPV$_{10\%}$	M$	562	768	1,171
PT-IRR	%	32	38	48
PT payback period	Production years	2.2	1.8	1.4

The break-even U_3O_8 price for the project is US$31/lb. The simplified pre-tax economic analysis for Case B is shown in Tables 21.2.

Table 21.2: Case B Economic Model

ELEMENT	DESCRIPTION	UNIT	TOTAL	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
MATERIAL SCHEDULE																			
Total Mining	**Mined Mineralization**	**Tonnes**	**732,000**					70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	28,000	
	Grade	**%**	**0.00**					3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	
	Total Mined lb	**Mlb U_3O_8**	**53**					5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	2.0	
Processing	Daily Mill Feed	Ore t/day	**193**					193	193	193	193	193	193	193	193	193	193	193	
	Mill feed	Tonnes	**732,000**					70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	70,400	28,000	
	Mill head grade	% U_3O_8	**3.30**					3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	
		Mlb U_3O_8	**53**					5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	2.0	
PLANT RECOVERY																			
Recovery	Mill recovery	%	**97.7%**					97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	97.7%	
Metal Production	U_3O_8 from mill	Mlb U_3O_8	**52**					5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	2.0	
NET SMELTER RETURN																			
Metal Prices																			
	U_3O_8 Price	US$/lb						**70**	70	70	70	70	70	70	70	70	70	70	
	Exchange rate	C$:US$						**1.05**	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	
	U_3O_8 Price	C$/lb	**74**					74	74	74	74	74	74	74	74	74	74	74	
Payable Metal	Percent payable U_3O_8	%	**100%**					100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	
	Payable U_3O_8	Mlb U_3O_8	**52.0**					5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	2.0	
Gross Income pre-royalties		**M$**	**3,824**					**368**	**368**	**368**	**368**	**368**	**368**	**368**	**368**	**368**	**368**	**146**	
		$/kg U3O8						**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	**162.04**	
Royalties																			
4%	Basic Sask. Uranium Royalty	M$	**153**					15	15	15	15	15	15	15	15	15	15	6	
	Tier I Sask. Uranium Royalty	M$						19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	
	Tier II Sask. Uranium Royalty	M$						12.41	12.41	19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	19.58	
	Tier III Sask. Uranium Royalty	M$						-	-	83.70	83.70	83.70	83.70	83.70	83.70	83.70	83.70	83.70	
Royalties	Basic + Tiered Royalties	M$	**463**					**20.2**	**20.2**	**50.3**	**50.3**	**50.3**	**50.3**	**50.3**	**50.3**	**50.3**	**50.3**	**20.0**	
Gross Income From Mining		**M$**	**3,361**					**348**	**348**	**317**	**317**	**317**	**317**	**317**	**317**	**317**	**317**	**126**	
OPERATING COST																			
	UG mining cost	M$	**308**					28.7	28.7	28.7	28.7	28.7	29.7	30.7	30.7	30.7	30.7	12.2	
	Processing OPEX	M$	**351**					33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	13.4	
	G&A	M$	**92**					8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	3.5	
	TOTAL OPEX	**M$**	**751**					**71**	**71**	**71**	**71**	**71**	**72**	**73**	**73**	**73**	**73**	**29**	
	Sensitized TOTAL OPEX	**M$**	**751**					**71**	**71**	**71**	**71**	**71**	**72**	**73**	**73**	**73**	**73**	**29**	
	UG Mining Unit OPEX	$/t ore mined	**421**					407.47	407.47	407.47	407.47	407.47	421.37	435.83	435.83	435.83	435.83	437.38	
	Processing OPEX	$/t milled	**480**					480.00	480.00	480.00	480.00	480.00	480.00	480.00	480.00	480.00	480.00	480.00	
	G&A	$/t milled	**126**					125.55	125.55	125.55	125.55	125.55	125.55	125.55	125.55	125.55	125.55	125.55	
	Unit OPEX	**$/t milled**	**1,026**					1013.02	1013.02	1013.02	1013.02	1013.02	1026.92	1041.38	1041.38	1041.38	1041.38	1042.93	
	Unit OPEX	**$/lb U3O8**	**14.44**					14.25	14.25	14.25	14.25	14.25	14.45	14.65	14.65	14.65	14.65	14.67	0.00
NET OPERATING INCOME																			
	NET OPERATING INCOME	**M$**	**2,610**			**-**	**-**	**276**	**276**	**246**	**246**	**246**	**245**	**244**	**244**	**244**	**244**	**97**	
CAPITAL COST																			
Underground	UG Mine Dev. & Equipment	M$	**159**	10.4	14.0	21.8	48.9	-	0.0	0.0	16.3	35.6	11.0	0.0	1.1	0.0	0.0	-	
Process	Process Plant	M$	**172**		17.2	86.0	68.8												
Infrastructure	Access Road	M$	**2**	2.4															
	Power Line	M$	**3**	2.5															
	Tailings Management Facility	M$	**31**				29.4				0.7			0.7					
	Camp and facilities	M$	**7**	5.0	2.0														
	Buildings and other facilities	M$	**10**			5.0	5.0												
Owners Costs	Other	M$	**8**	2.0	2.0	2.0	2.0												
EPCM	EPCM	M$	**48**	3.0	5.0	16.9	22.8												
Closure	Closure	M$	**14**																14.2
Working	Working capital	M$	**-**					23.8							- 23.8	-	-	-	
	Capital cost w/o contingency	**M$**	**454**	**25.4**	**40.2**	**131.7**	**176.9**	**23.8**	**0.0**	**0.0**	**17.0**	**35.6**	**11.0**	**0.7**	**- 22.7**	**0.0**	**0.0**	**-**	**14.2**
Contingency	Contingency	%	**25**	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
	Contingency	M$	**113**	6.3	10.1	32.9	44.2	-	0.0	0.0	4.2	8.9	2.8	0.2	0.3	0.0	0.0	-	3.6
	TOTAL CAPITAL COST	**M$**	**567**	**31.7**	**50.3**	**164.6**	**221.1**	**23.8**	**0.0**	**0.0**	**21.2**	**44.4**	**13.8**	**0.9**	**- 22.4**	**0.0**	**0.0**	**-**	**17.8**
EBITDA			**-**																
	EBITDA	**M$**	**2,043**	**-32**	**-50**	**-165**	**-221**	**253**	**276**	**246**	**225**	**202**	**231**	**243**	**267**	**244**	**244**	**97**	**-18**
7%	Annual Discounted EBITDA	M$	**1,025**	-32	-47	-144	-180	193	197	164	140	117	126	124	127	108	101	38	-6
	Cumulative EBITDA	M$		-32	-82	-247	-468	-215	61	307	532	734	965	1208	1475	1719	1963	2060	2043
	Cumulative Discounted EBITDA	M$		-32	-79	-222	-403	-210	-13	151	291	408	534	658	784	893	994	1032	1025

21.2 Sensitivities

Sensitivity analyses were done for all cases by individually modifying the capital cost, operating cost, metal price and grade up and down by 20% to show the sensitivity of the EBIT $NPV_{7\%}$. The results show that the project is very robust. Like most mining projects, the project is most sensitive to commodity price and mill feed grade. For Case A, a 20% increase in U_3O_8 price leads to a 40% increase in PT-$NPV_{7\%}$ from $769M to $1,076M. A change in grade by 20% has a similar effect on PT-$NPV_{7\%}$. The converse occurs if the metal price drops by 20%, the PT-$NPV_{7\%}$ drops from $769M to $443M.

Capital costs are the next most sensitive parameter. A 20% increase in capital costs in Case A reduces the PT-$NPV_{7\%}$ by $93M from $769M to $676M or -12%. In Case B, because of its higher base value, a 20% increase in capital costs only reduces the PT-$NPV_{7\%}$ by 9%.

Operating and capital cost sensitivities are very similar. A decrease in the PT-$NPV_{7\%}$ of $86M (-11%) occurs if operating costs are 20% higher than estimated in the PEA.

A summary of the sensitivity analysis is shown in Table 21.3 and Figures 21.1 to 21.3.

Table 21.3: Sensitivity Analysis Results

Case	Variable	PT-$NPV_{7\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	Capital Cost	862	769	676
	Operating Cost	856	769	683
	Metal Price	443	769	1,076
	Grade	423	769	1,096
Case B	Capital Cost	1,118	1,025	932
	Operating Cost	1,111	1,025	939
	Metal Price	648	1,025	1,377
	Grade	647	1,025	1,400
Case C	Capital Cost	1,620	1,527	1,434
	Operating Cost	1,613	1,527	1,441
	Metal Price	1,076	1,527	1,978
	Grade	1,053	1,527	2,001

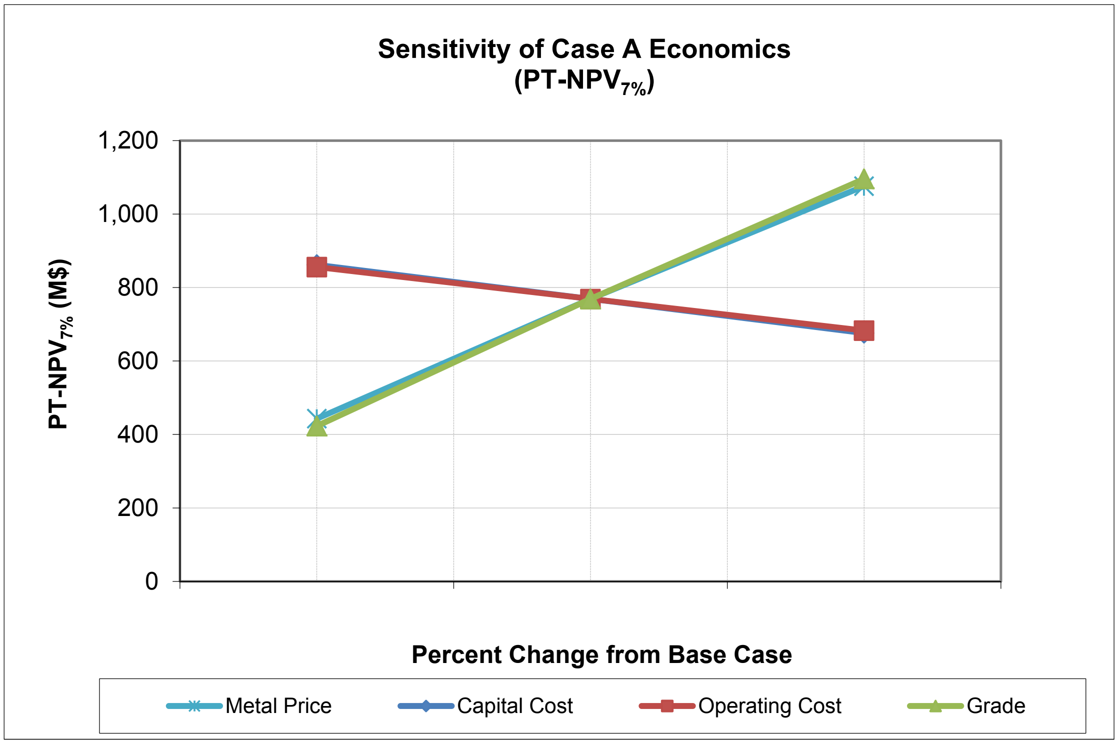


Figure 21.1: Case A Sensitivity Results

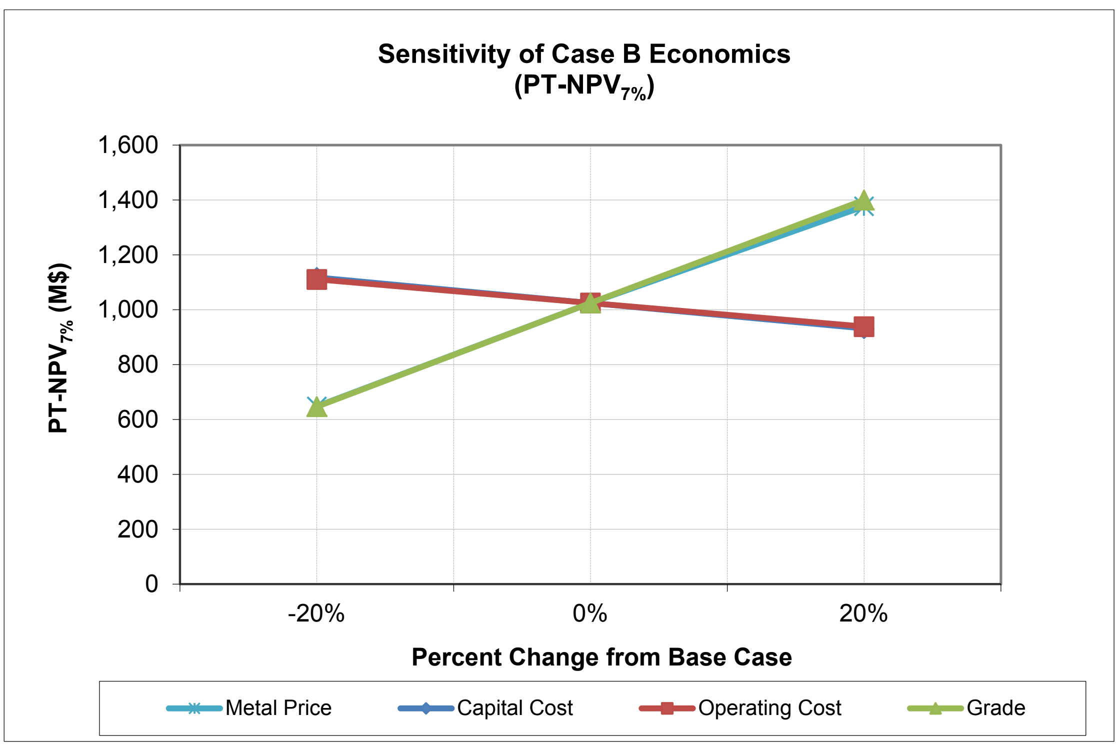


Figure 21.2: Case B Sensitivity Results

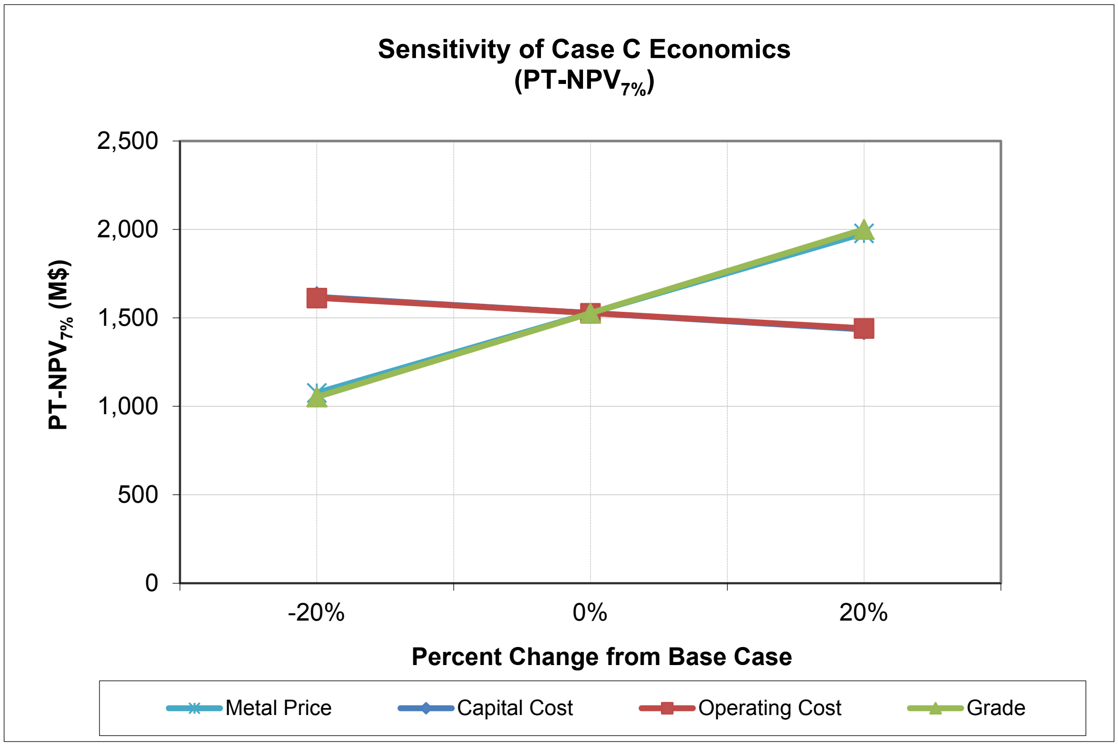


Figure 21.3: Case C Sensitivity Results

21.3 Mine Life

The mine production life at the Roughrider project encompassing the West and East deposits, is approximately 10.5 years.

There is potential to extend the mine life if further exploration targets, particularly the Far East Zone, can be upgraded to mineral resources and mineable tonnes.

21.4 Payback Period

The capital payback period for the project, based on the economic analysis assumptions is as shown in Table 21.4.

Table 21.4: Payback Period by Case

Case	Payback Period (Production Years)
Case A	2.2
Case B	1.8
Case C	1.4

21.5 Taxes and Royalties

The economic analysis performed in this study does not include taxes but does include royalties. Indicative Federal and Provincial taxes to which the project would be subject are summarized in the following section (reference: CostMine - InfoMine USA, Inc. and adapted as appropriate).

21.5.1 Federal Taxes

Income Tax

The general federal tax rate for Canadian-controlled private corporations (CCPCs) is 16.5% and is scheduled to be reduced to 15% as of January 1, 2012.

In general, the tax regulations shown here apply equally to domestic or foreign firms operating in Canada. Alternatively, a foreign firm can establish a Canadian corporation to conduct business in Canada. In general terms, federal taxable income for a mining company is defined as mining revenue less the following deductions:

- Operating costs;
- Capital cost allowance (CCA);
- Resource allowance;
- Canadian exploration expense (CEE);
- Cumulative Canadian development expense (CCDE);
- Interest expense; and
- Crown royalties and provincial mining taxes paid.

Goods and Services Tax

The GST is a tax paid on most goods and services sold or provided in Canada. The GST rate for Saskatchewan is 5%.

21.5.2 Provincial Taxes

Corporate Income Tax

Saskatchewan's corporate tax is levied as a percentage of the share of a corporation's federally defined taxable income that is allocated to the Province. Saskatchewan's general tax rate on corporate taxable income is 12%.

Capital Tax

Saskatchewan resource producers are subject to a capital tax surcharge.

Capital Tax Surcharge

The capital tax charged to qualifying potash, uranium, coal, oil and natural gas producers is 3.0%.

Provincial Sales Tax

The PST in Saskatchewan is 5%.

Property Tax

In Saskatchewan, property values are updated every four years. The Saskatchewan Assessment Management Agency (SAMA) conducts a full revaluation of all properties in the province to co-ordinate with a new base date. The 2009 property assessment system will move to a more flexible, results-focused market value assessment system which is essentially the same in all other Canadian provinces.

Three generally accepted appraisal techniques used to value property in a market value assessment system include the cost approach, the sales comparison approach and the property income (rental) approach. Of these, the only method currently allowed in Saskatchewan for valuing commercial properties is the cost approach which will continue to be used in smaller municipalities and for specific property types.

21.5.3 Uranium Royalties

In 2001, Saskatchewan introduced a uranium royalty structure governing all uranium production in the province. The structure is divided into two separate royalties, a Basic Royalty and a Tiered Royalty, that are computed separately. The two sums calculated are added to determine the total royalties due. The royalties are calculated for all operations of each firm rather than on a project by project basis. Following are the main features of the royalty structure.

Basic Royalty

The basic uranium royalty rate is equal to 5% of the value of gross uranium sales less transportation costs from mine to point of sale. The Saskatchewan Resource Credit, which is one percent of gross sales, is credited against the Basic Royalty thus the effective rate is 4%

Tiered Royalty

The Tiered Royalty is based upon unit gross sales values, with progressively higher royalty rates for increasing, realized uranium prices per pound. There is a minimum below which no royalty applies. The royalty is determined for sales falling within each of the increments defined. The rates are 6, 10, and 15%. When combined with the Basic Royalty, the total rate is 4, 10, 14, and 19% of gross revenues for sales in the price bracket of each tier. These prices are indexed for inflation from 1998. The index used for this PEA was 1.3056.

Table 21.5: Saskatchewan Uranium Royalty Structure

Price Bracket	Base U_3O_8 Price Range ($/kg)	Indexed U_3O_8 Price Range ($/kg)	Tax Rate (%)	Total Tax Rate[1] (%)
Tier One	30.00 to 45.00	39.17 to 58.75	6	10
Tier Two	45.00 to 60.00	58.75 to 78.34	10	14
Tier Three	> 60.00	> 78.34	15	19

[1] The total tax includes both Basic and Tiered royalties

A cumulative capital recovery bank is established for each producer in which a prescribed investment allowance is calculated in $/kilogram capacity. The bank is used to reduce the amount of revenues subject to the Tiered Royalty. The prescribed amount for new mills is $80/kg, mill expansions $50/kg, surface mines $45/kg, and underground mines $60/kg U_3O_8 capacity. These amounts are indexed for inflation and values of $104.45/kg and $78.34/kg were used for new mill and UG mine capital recovery bank allowances.

No more than 50% of the capital recovery bank total may be used to reduce revenues in a given year. After deduction of any capital recovery bank amounts, small producers, producing less than 2 million pounds per annum, are allowed a credit of $750,000, indexed for inflation from 1998. This additional deduction compensates for the increase in royalties under the new structure. Producers who had not applied for this credit prior to March 31, 2002 are not eligible for this additional deduction.

The royalty structure is undergoing review at the time of writing this report and it is unknown what the future structure will be.

The total royalty payments calculated in the PEA economic model are shown in Table 21.6.

Table 21.6: LOM Royalty Payment Estimates by Case

Case	U_3O_8 Price ($/lb U_3O_8)	LOM Royalty Estimate ($M)
Case A	60	366
Case B	70	463
Case C	90	669

22 Adjacent Properties

22.1 J and J East Zones

Adjacent property to the Roughrider Project is the group of Waterbury Lake Project, consisting of 13 mineral dispositions (40,256 ha) held by the Waterbury Lake Uranium Corporation (“WLUC”), a Limited Partnership owned by Fission Energy Corp. (50%) and Korea Waterbury Uranium Corp. (50%).

Disposition S-107370 hosts the J and J-East Zones. No NI43-101 technical reports are available for the J or J-East Zones. The following summary is compiled from numerous Fission Energy news releases and website, and a single abstract (McElroy 2010).

J Zone was discovered in winter 2010 with drillhole WAT10-063A, which intersected anomalous radioactivity immediately at the unconformity; the drillhole extended into the basement for 29.0 m and returned a composited core length interval of 10.5 m at 1.91% U_3O_8 (source: Fission Energy Corp. website). The best drillhole to date is WAT11-13, which intersected 7.84 % U_3O_8 over 14.5 m.

To date, the WLUC has intersected mineralization in 88 core metres. boreholes have defined the uranium mineralization over an area of approximately 578 m by up to 50 m.

Uranium mineralization occurs primarily at the unconformity, but may extend as a keel into the basement, immediately below the unconformity (McElroy 2010). Drill intersections in the J-Zone are characterized by a broad continuous zone of alteration extending from several metres above the unconformity to greater than 25.0 m below the unconformity, with uranium occurring within this altered system.

J Zone East, located approximately 15.0 m to west of the claim boundary is currently defined by three boreholes. The best intersection to date is Hole WAT10-102, which intersected a core length interval of 8.50 m grading 0.38% U_3O_8 (220.00 m to 228.50 metres) and 3.5 m of 0.67% U_3O_8 (233.00 m to 236.50 m) (source: Fission Energy Corp. website). The uranium mineralization is hosted in basement rock and is similar to that found at the Roughrider Uranium Deposit. The J-East Zone may represent an extension of the Roughrider Uranium Deposit.

No mineral resources have been declared for either J Zone or J Zone East zones.

23 Other Relevant Data and Information

23.1 Geotechnical Considerations

23.2 Geotechnical Considerations

A preliminary geotechnical evaluation has been conducted to assess and characterize the rock mass of the West and East Zones for underground mining. Based on this evaluation, input recommendations for mine design have been provided at a PEA level and recommendations have been made as to what will be required to move the geotechnical level of understanding to a Pre-feasibility and Feasibility level. A summary of the geotechnical evaluation is presented in this section.

This evaluation has been based on a limited geotechnical dataset as provided by Hathor including:

- Drill logs for 339 drillholes (ranging from MWNE-07-001 to MWNE-10-229) with the following parameters:
  - Lithology;
  - Total Core Recovery (TCR %);
  - Friability;
  - Fracture Frequency per meter (FF/m), and;
  - Clay alteration.
- Core photos for selected drillholes in the East and West zones (19 and 20 drillholes respectively). Photographs were reviewed for an interval approximately 40 m above the unconformity to the end of hole.

From the available data sources, drillholes were selected (based on data availability) by SRK as a basis for the more detailed evaluation. Figure 23.1 shows coverage of the reviewed drillholes for the East and West Zones.

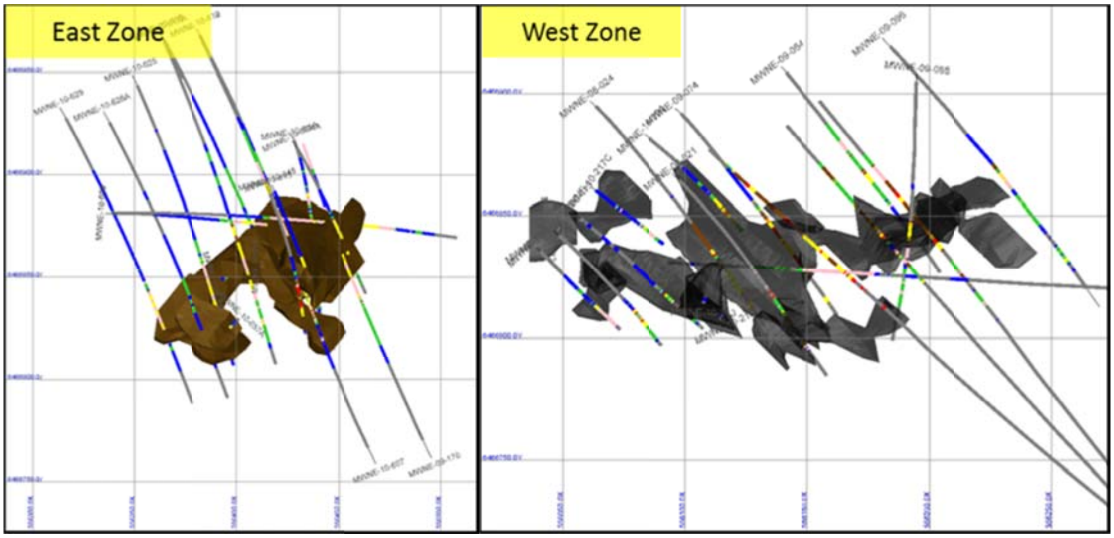


Figure 23.1: Plan View with Drillhole Coverage for Geotechnical Review

Mineralization is hosted beneath the regional unconformity within the basement meta-sediments and granitic-gneiss’. Based on SRK’s experience of uranium deposits within the Athabasca Basin, clay alteration (within basement units) and friability (primarily within sandstone units above the unconformity) are key indicators of poor ground conditions, and are typically most intense around the mineralized zones and major fault structures. These parameters have been used where possible to assist in the geotechnical characterization of the Rough Rider project.

In order to complete a qualitative assessment of the rockmass, selected core photographs were used to visually estimate a Rock Mass Rating (“RMR”). Completing this exercise provided an understanding of the likely ground conditions and enabled definition of preliminary geotechnical domains. The range of RMR values used for the interpretation were:

- ≤30 very poor to poor rock;
- 31-40 poor rock;
- 41-60 fair rock;
- 61-80 good rock, and;
- 81-100 (very good rock).

Figure 23.2 shows core photographs of RMR values ≤30.

Figure 23.2: Example of Weak Ground with RMR ≤30

23.2.1 Structural Model

A regional scale structural review has been completed by SRK for exploration purposes. The deliverables from the study included:

- Regional scale 2D fault interpretation based on magnetic data (covering a large area around Roughrider);
- District scale 3D fault interpretation based on 2.5 D seismic data and validated where possible with drillhole data. Note this also covers a wide area around Roughrider;
- A drillhole validation table listing the faults, drillholes where they have been intersected, and the fault characteristics; and
- A snapshot of the 3D fault model showing the location of the Roughrider project and fault labels.

All modelling and interpretation was completed in UTM NAD83 Zone 13 North. Although the model was not specifically constructed to a mine scale, all faults were utilized as part of the geotechnical evaluation to assist in the locating of mine infrastructure and understanding potential structural patterns at a more local scale (Figure 23.3).

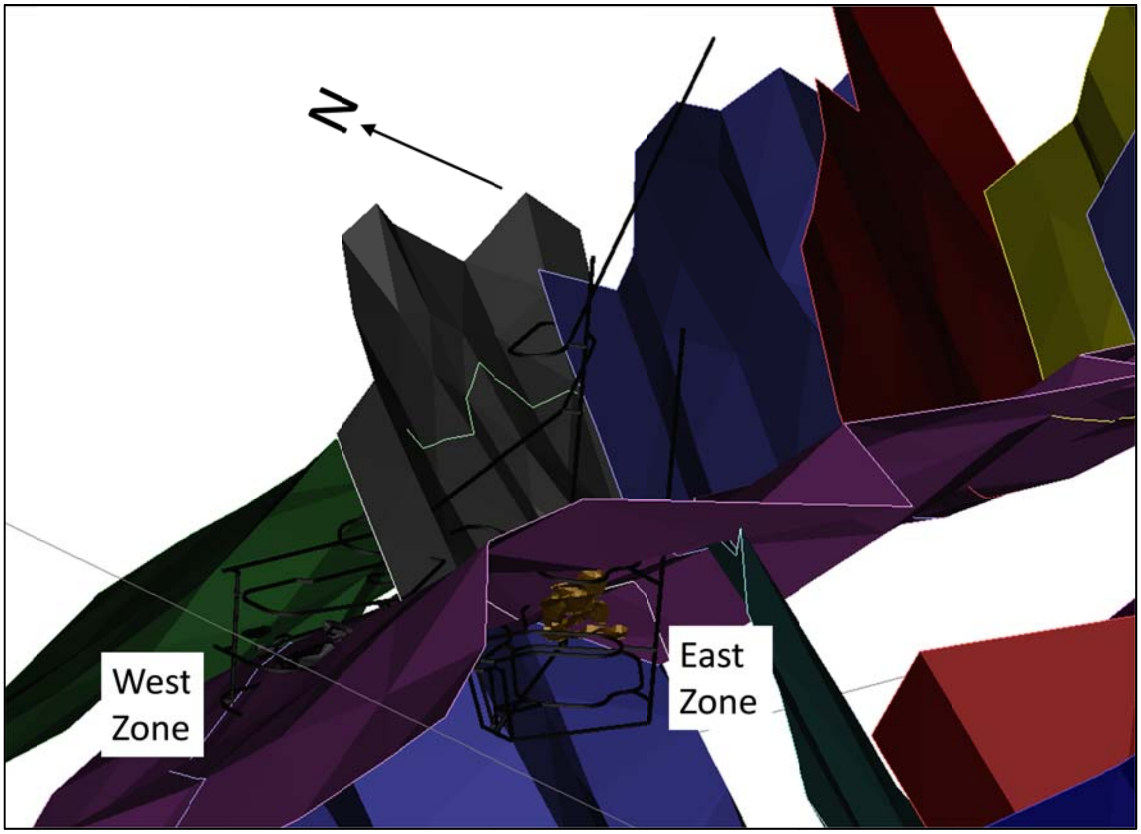


Figure 23.3: Regional Structural Model showing West and East Zones

23.2.2 Geotechnical Domains

The primary geotechnical domains are based around lithological boundaries as defined by Hathor. Using the available geotechnical data, these have been further broken down to spatially constrain rock mass variability.

High Risk Domains

The available geotechnical data have been imported and reviewed using Gemcom GEMSTM software. Cross-sections (four at West zone and five at East zone) have been selected through each zone to determine spatial patterns of low strength ground conditions and assist with geotechnical domain selection. Data within a 20-25 m corridor of the cross section has been used to define the likely areas considered to represent high geotechnical risk to mining around the high grade shells. Based on core photo reviews of clay alteration and visual estimation of rock mass quality, an RMR≤30 has been used to define these High Risk Domains (Figure 23.4). Figure 23.5 shows poor ground domains based on clay alteration intensity values of ≥4.

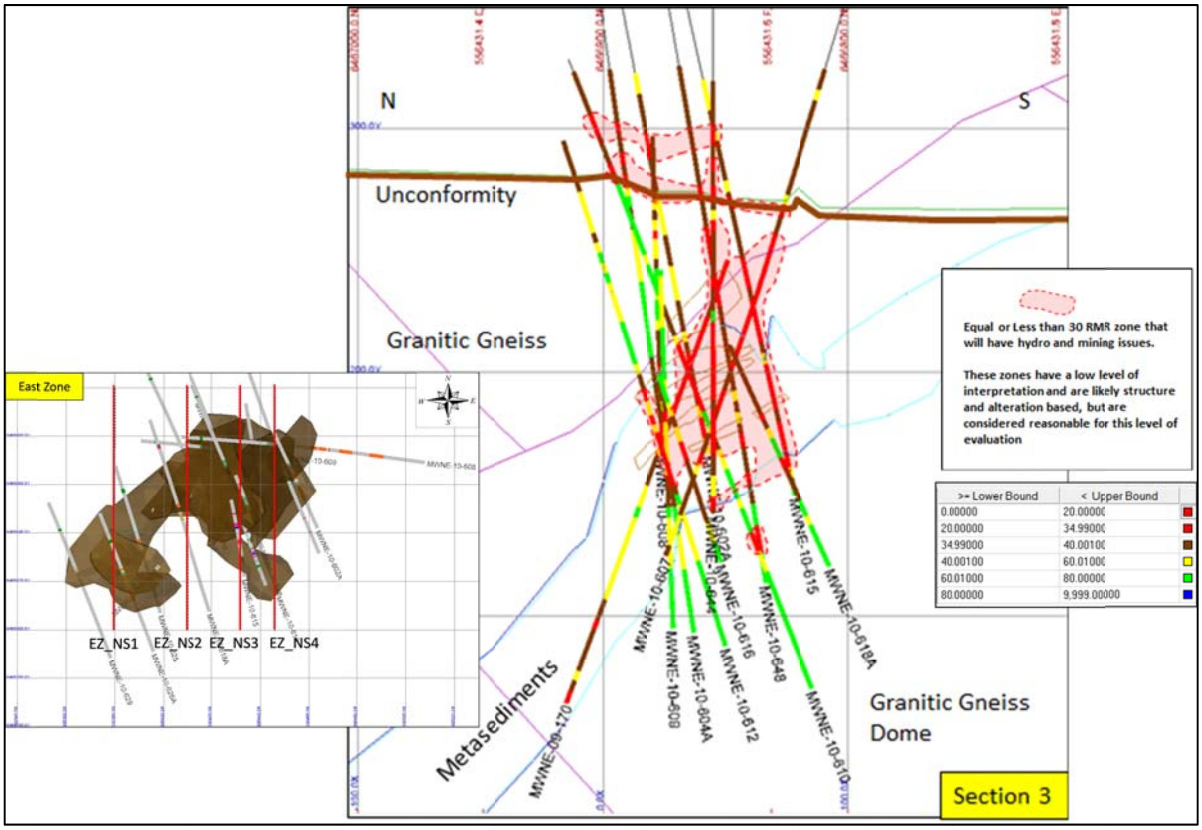


Figure 23.4: **Example Cross Section through the East zone showing Interpreted High Risk Geotechnical Domains based on RMR**

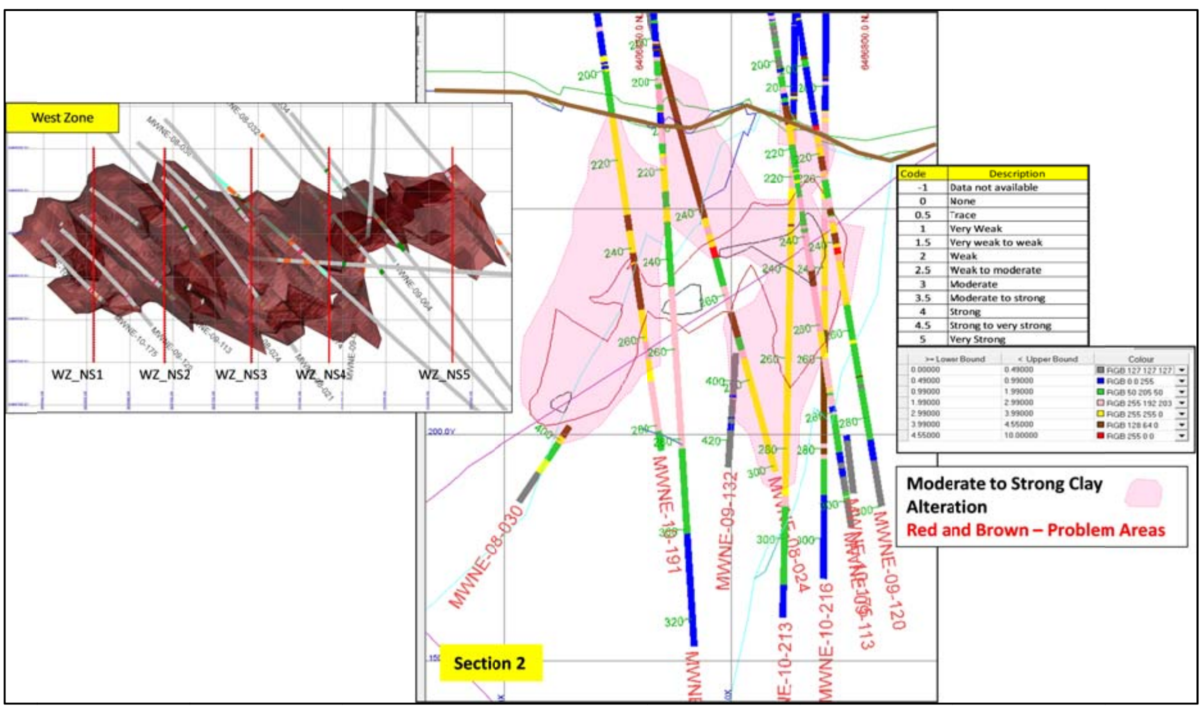


Figure 23.5: **Example Cross Section through the West Zone showing Interpreted High Risk Geotechnical Domains based on Clay Alteration Intensity**

Sandstone Domain

The Sandstone Domain contains the sandstone/conglomerate units above the Unconformity. Variability is anticipated to be low with generally Fair to Good rock mass quality prevailing (intact rock strength estimated at 60 to 120 MPa). Weaker and more friable zones should be expected in close proximity to major structures and mineralization (Figure 23.6). A complete assessment of the sandstone domain should be completed for all areas proximal to planned development.

Figure 23.6: **Fair to Good Rock Mass Conditions in the Sandstone Domain**

Unconformity Domain

The Unconformity Domain encompasses a zone of ground approximately 20 m either side of the regional unconformity surface where ground conditions are interpreted to exhibit a wider variability compared to the surrounding Sandstone and Basement Domains. An increased frequency of core loss, percent clay, and rubble is observed in all lithological units (Figure 23.8).

Figure 23.7: Variable ground conditions within the Unconformity domain

Basement Domain

The Basement domain encompasses the rock mass outside the interpreted High Risk Domains including meta-sediments, and granitic gneiss. Similar to the Sandstone Domain, variability is expected to be low, with predominantly Fair to Good rock mass conditions with rock strength in the range 80 to 150 MPa (Figure 23.7). Weaker zones should be expected in close proximity to major structures.

Figure 23.8: Fair to Good rock mass conditions in the Basement domain (distal to mineralization and major structures)

23.2.3 Excavation Design and Support Requirements

Based on the anticipated ground conditions, a lateral development heading of 5 m by 5 m is considered appropriate. Spans that are opened over this dimension should be supported with pattern cable bolts in addition to standard support described below.

Due to the likely presence of water, some level of cover grouting will be completed for all lateral development within sandstone, and to approximately 20 m vertical depth beneath the unconformity.

Lateral Development

Lateral infrastructure for access to the mining horizons has been designed to consider the High Risk Domains identified. The key objective of any development headings should be to avoid poor and weak ground conditions where possible. As such, it is recommended that development should be undertaken only after a thorough definition of the geotechnical domains within the proposed development area and completed under continuous probe and cover drilling at regular intervals along the main development headings.

Sandstone and Basement Domains: 2.4 m fully grouted resin rebar on a 1.2 m by 1.2m spacing with #7 gauge weld wire mesh. Plain shotcrete (75 mm – 100 mm) will be required in weaker zones. Spacing of rebar should be reduced based on ground conditions at the face. In particularly weak areas such as fault intersections spiling, early shotcrete application, and steel set support may be required

Production Mining

The basement mineralization contains significant variability within the rock mass characteristics. At this stage it is anticipated that the mineralization will not require ground freezing to utilize the selected mining method. Ground freezing is only required to seal off the mineralized zone from the hydrogeology threat from the overlying sandstones. The high grade of the deposit requires that a remote mining method be used such as raisebore mining. The diameter of the raisebore should be selected based on the lowest anticipated rock mass strength.

23.2.4 Geotechnical Considerations for Project Advancement

Major structures

At this stage of the project, a mine scale structural interpretation has not been completed. Structural patterns identified as part of the regional exploration interpretation should be further interrogated through diamond drilling, with a mine scale interpretation and the production of updated 3D wireframes completed by a Structural Geologist. This is critical to define the hydrogeological risk and help define the likely areas of poor ground conditions.

Material Properties

A detailed strength testing program should be completed to understand the range of strengths in each geotechnical domain.

As the weaker materials (within faults, alteration products and mineralization) will affect ground freezing performance, and a suite of physical properties and mineralogy testing should be completed for the various units.

Water

Cover grouting will be required to handle the potential high water pressures and volumes anticipated when mining within the Athabasca Sandstone. Additional investigation should be completed to review the need for cover grouting in the basement units, targeting potential connectivity through major structures from the Sandstone. Additional investigation should be completed to relate structural feature sets to the presence of water, and potential water inflows.

A substantial investigation program will be required to fully evaluate the option of using a decline rather than a shaft to access the mineralized zone.

Consideration should be paid to the groundwater quality (particularly salinity) within areas that will require ground freezing.

Ground Freezing

At this stage in the project, it has been assumed that the basement units distal to the mineralized zone are not hydraulically connected to the Athabasca Sandstone. This assumption will need to be re-considered once a better understanding of the mine scale fault structures and the potential hydrogeological connectivity has been established. The presence of water in large volumes or under high pressure will affect the quality of the planned ground freezing and grouting programs.

Additional characterization and evaluation of the mineralised rock mass is required to further validate the assumption that raisebore stability can be achieved without the need to freeze the mineralized zone.

Ground temperature will have an effect on the quality of the planned ground freezing and the collection and review of ground temperature data from any possible source should be completed.

23.3 Hydrogeological Considerations

SRK is not aware of any hydrogeological or hydrologic data for the specific site. Subsurface investigations have focused on exploration drilling, which provide input on the potential distribution of geologic units or hydrogeological domains that could affect inflows or water management, but not specific information on hydrogeological parameters. Hydrologic monitoring (i.e., stream flow monitoring, climatic monitoring) was not available at the time of this review.

Key sensitivities for mining are: 1. The workings are below, or in close proximity to a lake, and 2. Geologic units being mined have potential to be highly permeable and water bearing, as shown by experiences at other nearby sites.

Underground Inflow mitigation measures include the decline cover grouting and the mine area freeze shells. Once in place, these measures will limit, though not necessarily stop, inflow. In general, groundwater inflows will be controlled by the hydraulic conductivity of units intersected by development or workings and the conductivity reduction provided by the mitigation measures. While site-specific information is not available, experience at other mines in the region suggests that inflows can be high if not managed.

The tailings storage facility ("TSF") is assumed to be a purpose-built pit with a pervious surround. The TSF will be excavated into a side slope. Key hydrogeological issues that should be considered include: 1. Requirement for dewatering in slope design, 2. Functionality of the pervious surround (i.e., relative hydraulic conductivity between surround material and natural material), and 3. Decommissioning and reclamation.

The design for the pervious surround concept will be reliant on the hydraulic conductivity of materials surrounding the TSF and the final water level within the TSF. Any TSF design and permitting will require an eco-risk assessment be completed; the design will have to minimize the potential for significant environmental effects. In particular, the surround design will need to consider water management during construction, potential pervious surround effectiveness issues during operations and management of groundwater seepage following closure.

From a risk perspective, designs can likely accommodate almost any conditions, but the time required to permit such a facility can be significant and must account for the timeframe over which the necessary environmental studies will be conducted. Considering the current uncertainty in regards to subsurface conditions in the potential TSF area, the greatest risk is the potential for permitting delays related to proving suitability of location and design.

It is likely that diversion ditches will be constructed to direct surface water away from the TSF thereby minimize the surface water contribution to the pervious material underlying the tailings pond. Minimizing the water contribution will reduce the thickness of the pervious material needed to convey water below the tailings pond.

24 Interpretation and Conclusions

Exploration work by Hathor is professionally managed and uses procedures meeting or exceeding generally accepted industry best practices. After review, SRK is of the opinion that the exploration data collected by Hathor are sufficiently reliable to interpret with confidence the boundaries of the uranium mineralization for the Roughrider Uranium Deposit and support evaluation and classification of mineral resources in accordance with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" and CIM "Definition Standards for Mineral Resources and Mineral Reserves" guidelines.

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Roughrider deposit. There is adequate geological and other pertinent data available to generate a PEA. No fatal flaws were discovered during the study.

Based on current knowledge and assumptions, the results of this study show that the project is economically robust.

SRK recommends that the project be advanced to the next level of study based pending additional mineral resource definition drilling and engineering information acquisition (geotech, hydrogeology, geochemistry, metallurgy, etc.).

25 Risks

As with most preliminary studies, there are a large amount of technical data and information that is not yet available for the Roughrider project, which requires the use of assumptions and estimates based on experience, reference projects, etc. This lack of information may lead to significant changes in mine planning and cost estimates when the project is advanced to the next study level and detailed engineering studies are conducted. Tables 25.1 and 25.2 describe the main potential internal and external risks to the project as currently known.

Table 25.1: Internal Risks

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Resource Estimate	All of the East deposit mineral resources are Inferred and, as such, carry a high level of geological risk. The ability to transform Inferred resources to Indicated and Measured is not guaranteed.	Inferred mineral resources that are not converted to Indicated or Measured would have to be left out of the FS and would negatively impact mine life and project economics.	Further drilling is planned in 2012 top attempt to convert Inferred resources into Indicated or Measured.
Process Recoveries	A reduction in metal recoveries would have a significant impact on project economics. The East zone mineralized material has not had metallurgical tests conducted on it and, for the purpose of this PEA, was assumed to have similar mineral characteristics as the West Zone.	A reduction in metal recovery of 1% would reduce the Case B PT-$NPV_{7\%}$ by about \$20M	Metallurgical testing is required, for the East and Far East Zones prior to the next level of study.
CAPEX and OPEX	Estimated CAPEX and OPEX costs are only preliminary and their variance would impact the economics of the project.	As shown in the sensitivity analysis, every 1% increase in CAPEX or OPEX decreases the PT-$NPV_{7\%}$ by about \$4.5M for Case B.	Further cost accuracy with the next level of study as well as the active investigation of potential cost-reduction measures
Permit Acquisition	The ability to secure all permits from regulators is of paramount importance to the project.	The project would not be allowed to be built or operate if the required permits are not granted.	The development of a close relationship with the communities and government along with a thorough EIA and a project design that gives appropriate consideration to the environment and local people is required.
Hydrogeology and UG Water Control	The proposed Roughrider Mine would be located approximately 220 m below a lake and only	Failure to fully understand the hydrogeological characteristics of the proposed mine area and	Further hydrogeology work is required to define the local conductivity of the rocks and geological structures in the mine area

	metres below the unconformity zone, a known water bearing feature. The ability to control the inflow of ground water into the mine in the PEA relies on the assumption that mine development can control water with grout cover and that stoping areas can be mined using single layer freeze walls.	develop robust plans to mitigate the ground water inflow could lead to mine flooding, delays in mine development or both. If double-layer freeze walls are required and the walls are required to be maintained deeper into the basement rock, the mine CAPEX would increase by roughly $ 50M.	and is to be pursued in subsequent field programs. Depending on the outcome of the further studies the freezing and grouting plans for the mine may have to be modified. All exploration drillholes have been cemented to mitigate water conductivity.
Rock Mechanics	The proposed mining methods and mine designs are based on assumptions made with minimal geotechnical information. The geotechnical characteristics of the deposits would affect the mining method, dilution and extraction.	It the geotechnical attributes of the rocks are not conducive to raiseboring (i.e. they experience instability within the raisebore hole) then the mining method could be compromised or mining costs significantly increased and extraction decreased. If total LOM extraction is decreased by about 100,000 tonnes (14% of the total), the project PT-NPV$_{7\%}$ drops from $1,025M to $886M for Case B.	Further geotechnical work is required to define the conditions of the proposed mine area and are to be pursued with subsequent field programs.
Mine Access	The PEA assumes a ramp access through overlying sandstones into the basement rock. To date there are no access ramps in the Athabasca Basin that have access development through sandstone.	If an access decline is not practical through the overlying sandstone, a shaft, approximately 250 m deep would have to be sunk to provide mine access. This would increase mine development capital costs by potentially $10-$30M but may shorten the project construction schedule.	Detailed hydrogeological and geotechnical studies would be required to adequately characterize the proposed mine access conditions and are to be pursued with subsequent field programs.
Development Schedule	The development schedule may be aggressive if geotechnical or hydrogeological difficulties are encountered and the project could be delayed depending on several factors.	A change in schedule would alter the project economics by having a longer lead time until production.	Further field work is needed to better define the mining environment.
Mining Dilution and Extraction	The ability to extract mineralized material with controlled dilution and to the desired grade	The project is very sensitive to mill head grade. Waste dilution would add to treatment	Selective mining methods have been chosen for the UG deposits which actually rely on waste

	of 3.3% U_3O_8 is a potential risk.	costs per lb of U_3O_8 and have a negative impact on project economics. Given a constant tonnage, a reduction in mill head grade of only 0.1% U_3O_8 would lead to a drop in PT- $NPV_{7\%}$ of \$57M and a loss of 1.5 Mlb of U_3O_8.	dilution to bring the mill head grade to 3.3% U_3O_8.. Further geological definition of the deposits could help verify geologic modelling and improve mining confidence.
Tailings Management Facility ("TMF")	The suitability of the TMF in its current location and configuration is a risk as it is based on minimal data.	The design for the pervious surround concept will be reliant on the hydraulic conductivity of materials surrounding the TMF and the final water level within the TMF. Any TMF design and permitting will require an eco-risk assessment be completed; the design will have to minimize the potential for significant environmental effects. The time required to permit such a facility can be significant and must account for the timeframe over which the necessary environmental studies will be conducted.	Increased information from an extensive field program (with corresponding lab work) is required to determine the TMF suitability.
Mining Method	The raisebore mining method was selected based on the current knowledge of the characteristics of the deposits and experience at other similar deposits. In the West zone, the piloting of raisebore holes would be required to pass through the unconformity.	If the West and East deposits are not suitable for the raisebore mining method, the mine operating and capital costs as well as production capability would have to be re-estimated based on a new method of rock mass conditions. An alternate mining method may not yield the same production rate, dilution, extraction or flexibility and may have a significant impact on economics.	Greater understanding of the deposit geology, hydrogeology and rock mechanics are needed and are to be pursued with subsequent field programs.
Inclusion of Taxation and Financing Costs	No taxation or financing costs were included in the economic evaluation.	Taxation and financing costs will reduce the NPV of the project.	Include taxation and financing costs in the next level of study.

Table 25.2: External Risks

Risk	Explanation	Potential Outcome	Possible Risk Mitigation
U_3O_8 Price	Uranium price has a significant impact on the economic viability of the project.	In Case B, a 20% drop in uranium price (from US$70/lb to US$56/lb) produces a drop in PT-$NPV_{7\%}$ of almost $380M.	The early (pre-construction) establishment of a long-term uranium sales contract.
Electrical Power Supply	Based on discussions with SaskPower it was assumed that there is sufficient electric grid power near Points North to supply the needs of the Roughrider project.	Failure of adequate grid power might mean that the project would have to fund the additional infrastructure needed to power the project.	Further discussions and contractual power supply and tariff guarantees from SaskPower.
Taxation	Taxation was not taken into account in this PEA and will have a negative impact the NPV of the project	The project NPV will be negatively impacted by the inclusion of taxation. The extent of this impact is not known but the reader is advised to review the taxation section in this report for the general prevailing tax structure in the province.	The next level of study should take taxation into consideration.
Fuel, Power, Reagent and Labour Costs	The price of consumables are based on global economics and could impact the project if they rise substantially. The cost of labour is largely dictated by regional competition, which can lead to cost escalation.	An increase in costs could increase both operating and capital costs.	Long-term supply contracts may mitigate some of the consumable price uncertainty. The economic analysis suggests that the project is only moderately sensitive to input costs.
Cost of Capital (discount rate)	The selection of the discount rate can have a large impact on the project NPV, especially for projects with long pre-production periods.	A change in the discount rate from 7% to 10% decreases the project PT-NPV from $1,025M to $698M	The discount rate is set by the company based on its cost of capital.
Economic Analysis Term	The start of economic analysis for this PEA was selected at the beginning of the 4-year development and construction period. Prior to getting to the commencement of construction, it is estimated that an additional minimum of 4 years would be needs for engineering studies, permitting and financing. This equates to a best-case estimate of 8 years	If the economic analysis period was started from the date of this report, and assuming pre-development costs of $20M spread over a 4-year period (making the total pre-production period 8 years) the PT-$NPV_{7\%}$ would drop from $1,025M to $759M for Case B.	Further project timing refinements and sensitivities should be developed at the next level of study.

	to reach commercial production from the date of this report.		
Project Financing	The project will require extensive equity or bank financing and a joint venture with an experienced operating partner or purchase from a larger producing company (or a combination of the above).	The rate of project development will be dependent on the ability to secure financing.	Continued value-adding field work including additional resource development/definition and technical studies as well as developing a financing plan if the project continues to develop are needed
Hiring of Mining Contractor	The selection of an appropriate mining contractor, with Athabasca Basin experience, will be critical to success of the project.	Project timing, costs and overall success will be affected by the company's ability to retain a skilled mining contractor.	The early search for an experienced mining contractor is highly recommended.
Recruiting Experienced Professionals for the Development and Operating Teams	The selection of appropriate, experienced people for the project will be important to its success. The global demand for experienced minerals industry personnel is a concern.	Project timing, costs and overall success will be affected by the company's ability to assemble a skilled development and operating team.	The early search for an experienced workforce would be required along with appropriate compensation and benefits.

26 Opportunities

Opportunity	Explanation	Potential Impact
Exploration Potential to Increase Mineral Resources	The inclusion of the Far East Deposit and/or other exploration targets in the mineral resource estimate may increase the mineable tonnes and improve project economics.	
Toll Processing	The toll treatment of Roughrider mineralized rock at an existing (non-Hathor) processing plant in the region could add value to Hathor and the receiving processing plant.	Processing plants in the region are not currently operating at full capacity and if a toll treatment agreement was reached it could provide a benefit to Hathor by eliminating the need to spend capital on its own processing plant. The receiving plant could potentially benefit from treating Roughrider material by collecting a toll for processing, increasing throughput and therefore, reducing unit costs.

27 Recommendations

Based on the positive results of this preliminary analysis, SRK recommends that the project be taken to the next level of study. SRK recommends that a preliminary feasibility study ("PFS") be conducted but understands that Hathor wishes to proceed directly to a feasibility study ("FS"). A FS would require the following components:

Field Programs

- Definition drilling of the East and Far East Zones with the goal of taking them to Indicated or Measured classification when a revised estimate is done for the FS;
- Augmentation of the specific gravity database to define the spatial variability of the specific gravity in all resource domains.
- Metallurgical sampling and testing to a FS level;
- Geotechnical drilling of the West, East and Far East deposits, mine accesses and surface facilities;
- Hydrogeological drilling of the West, East and Far East deposits, mine accesses and surface facilities;
- Tailings Management Facility studies;
- Initiation of environmental baseline studies;
- Hydrology studies; and
- Geochemical sampling and testing.

FS Study

- Following the completion of the field program a FS would be completed.
- The cost of the field program and FS would be between $8M to $10M and would take a minimum of 12-18 months.

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29 Abbreviations and Acronyms

Distance	
μm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	km
” or in	inch
’ or ft	foot
Area	
ac	acre
ha	hectare
Time	
s	second
m or min	minute
h or hr	hour
d	day
y or yr	year
Volume	
l	litre
usg	US gallon
lcm	loose cubic metre
bcm	bank cubic metre
Mbcm	million bcm
Mass	
kg	kilogram
g	gram
t	metric tonne
Kt	kilotonne
lb	pound
Mt	megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
As	arsenic
Cu	copper
Fe	iron
Mo	molybdenum
Pb	lead
S	sulphur
U_3O_8	triuranium octoxide a constituent of “yellowcake”
U	uranium
Zn	zinc
Electricity	
kW	kilowatt
kWh	kilowatt hour
V	volt
W	watt
Ω	ohm
A	ampere

Unit Prefixes	
μ	micro (one millionth)
m	milli (one thousandth)
c	centi (one hundredth)
d	deci (one tenth)
k or K	kilo (one thousand)
M	Mega (one million)
G	Giga (one trillion)
Temperature	
^oC	degree Celsius (Centigrade)
^oF	degree Fahrenheit
Misc.	
Btu or BTU	British Thermal Unit
Ø	diameter
r	radius
hp	horsepower
s.g.	specific gravity
masl	metres above sea level
elev	elevation above sea level
Rates and Ratios	
p or /	per
mph	miles per hour
cfm	cubic feet per minute
usgpm	United States gallon per minute
tph	tonnes per hour
tpd	tonnes per day
mtpa	million tonnes per annum
ppm	parts per million
ppb	parts per billion
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	acid- base accounting
AP	acid potential
NP	neutralization potential
ML/ARD	metal leaching/ acid rock drainage
PAG	potentially acid generating
non-PAG	non-potentially acid generating
RC	reverse circulation
DD / DDH	diamond drill / diamond drillhole
IP	induced polarization
HL	heap leach
COG	cut off grade
NSR	net smelter return
NPV	net present value
LOM	life of mine
EBIT	earnings before interest and taxation
IRR	internal rate of return
DR	discount rate
PA	preliminary assessment
PFS	preliminary feasibility study
FS	feasibility study
Conversion Factors	
1 tonne	2,204.6 lb
1 troy ounce	31.1035

30 Signature Page

This technical report was written by the Qualified Persons listed below. The effective date of this technical report is September 13, 2011.

Qualified Person	Signature	Date
Gordon Doerksen, P.Eng	ORIGINAL SIGNED	October 26, 2011
Bruce Fielder, P.Eng.	ORIGINAL SIGNED	October 26, 2011
Iouri Iakovlev, P.Eng.	ORIGINAL SIGNED	October 26, 2011
David Keller, P.Geo.	ORIGINAL SIGNED	October 26, 2011
Mark Liskowich, P.Geo.	ORIGINAL SIGNED	October 26, 2011
Bruce Murphy, FSAIMM	ORIGINAL SIGNED	October 26, 2011
Cam Scott, P.Eng	ORIGINAL SIGNED	October 26, 2011

Reviewed by

ORIGINAL SIGNED

Mark Liskowich, P.Eng.

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

SRK Consulting (Canada) Inc.
Suite 2200 - 1066 West Hastings Street
Vancouver, BC V6E 3X2

T: +1.604.681.4196
F: +1.604.687.5532

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Gordon Doerksen, P.Eng.

I, Gordon Doerksen, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I am also a registered Professional Engineer in Yukon, Wyoming and Alaska. I am a member of the Canadian Institute of Mining and the Society of Mining Engineers of the AIME. I graduated with a BS (Mining) degree from Montana College of Mineral Science and Technology in May 1990.

I have been involved in mining since 1985 and have practised my profession continuously since 1990. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in Africa, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Roughrider Project site on June 27, 2011.

I am the responsible QP for the Executive Summary and Sections 1, 2, 14, 17, 18, 20.1.1, 20.2.1, 20.2.4, 21 and 24-30 of the "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am independent of Hathor Exploration Ltd. as independence is described by Section 1.5 of NI 43-101.

I have not previously been involved with the Roughrider Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL PROVINCE OF G. E. DOERKSEN # 32273 BRITISH COLUMBIA ENGINEER

Gordon Doerksen, P.Eng.

Dated: October 26, 2011

CERTIFICATE OF BRUCE C. FIELDER

Bruce C. Fielder, P.Eng.
Principal Process Engineer, Melis Engineering Ltd.
Suite 100, 2366 Avenue C North, Saskatoon SK Canada S7L 5X5
Tel: (306) 652-4084 Fax: (306)653-3779 Email: melis@sasktel.net

I, Bruce C. Fielder, am a Registered Professional Engineer in the Province of Saskatchewan, Registration No. 10309. I am Principal Process Engineer at Melis Engineering Ltd. with a work address of Suite 100, 2366 Avenue C North, Saskatoon, Saskatchewan, Canada.

1) I am a member of the Canadian Institute of Mining Metallurgy and Petroleum and I hold a Consulting Engineer designation with the Association of Professional Engineers and Geoscientists of Saskatchewan. I graduated from the University of Alberta with a BSc. Degree in Metallurgical Engineering in 1981.

2) I have practiced my profession continuously since 1981 and have been involved in: metallurgical testwork supervision, process engineering, preparation of process audits, scoping, pre-feasibility, and feasibility level studies, and mill operations for precious metals, base metals, uranium, rare earth elements and diamond projects worldwide.

3) I have read the definition of "*Qualified Person*" set out in "Standards and Guidelines for Valuation of Mineral Properties" (CIMVal Standards and Guidelines) published by the Canadian Institute of Mining, Metallurgy and Petroleum and National Instrument 43-101 and certify that, by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "*Qualified Person*" for the purposes of CIMVal Standards and Guidelines.

4) I served as the Qualified Person for Sections 12, 16, 20.1.3 and 20.2.3 of the report entitled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011. (The Report). The work was completed at the project site and in the Melis Engineering Ltd. office.

5) I visited the Roughrider Project site on June 27, 2011.

6) I have not had prior involvement with the property that is the subject of the Valuation Report.

7) As of the date of this certificate, to the best of my knowledge, information and belief, the metallurgical testwork, mineral processing, operating and capital cost estimation sections (Sections 12, 16, 20.1.3 and 20.2.3) of the Report contains all scientific and technical information that is required to be disclosed to make the metallurgical component of the Technical Report not misleading.

8) I am independent of the Issuer in accordance with the application of Section 1.5 of National Instrument 43-101.

9) I have read "Standards and Guidelines for Valuation of Mineral Properties" (CIMVal Standards and Guidelines) and National Instrument 43-101 and certify that the portions of the Report for which I served as a Qualified Person have been prepared in compliance with that Instrument.

Dated this 26th day of October, 2011.

Bruce C. Fielder

Bruce C. Fielder, P.Eng.

PROFESSIONAL ENGINEER
B.C. FIELDER
MEMBER 10309
YR. MO. DAY
SASKATCHEWAN

SRK Consulting (Canada) Inc.
Suite 2200 - 1066 West Hastings Street
Vancouver, BC V6E 3X2

T: +1.604.681.4196
F: +1.604.687.5532

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Iouri Iakovlev, P.Eng.

I, Iouri Iakovlev, am a Professional Engineer, employed as a Senior Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I am a graduate of the Siberian State Industrial University, Novokuznetsk, Russia (Mining Engineer, 1983).

I have practiced my profession for more than 20 years. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in North and South America, Africa, Europe and Asia.

As a result of my education, affiliation with a professional association and relevant work experience, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have not visited the Roughrider Project site. Information relayed by G. Doerksen.

I am the responsible QP for Sections 15,20.1.2,20.1.4,20.2.2 of the "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am independent of Hathor Exploration Ltd. as independence is described by Section 1.5 of NI 43-101.

I have not previously been involved with the Roughrider Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Iouri Iakovlev, P.Eng.

Dated: October 26, 2011

U.S. Offices:
Anchorage 907.677.3520
Denver 303.985.1333
Elko 775.753.4151
Fort Collins 970.407.8302
Reno 775.828.6800
Tucson 520.544.3688

Mexico Office:
Guadalupe,
Zacatecas
52.492.927.8982

Canadian Offices:
Saskatoon 306.955.4778
Sudbury 705.682.3270
Toronto 416.601.1445
Vancouver 604.681.4196
Yellowknife 867.873.8670

Group Offices:
Africa
Asia
Australia
Europe
North America
South America

SRK Toronto
2100 – 25 Adelaide Street East
Toronto, ON M5C 3A1

T: +1.416.601.1445
F: +1.416.601.9046

toronto@srk.com
www.srk.com

CERTIFICATE OF QULIAFIED PERSON

To accompany the technical report entitled: "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan," effective date of September 13, 2011..

I, G. David Keller, residing at 255 Richmond Street East, Toronto, Ontario do hereby certify that:

1) I am a Principal Resource Geologist with the firm of SRK Consulting (Canada) Inc. ("SRK") with an office at Suite 2100, 25, Adelaide Street East, Toronto, Ontario, Canada;
2) I am a graduate of the University of Calgary in 1986. I obtained a B.Sc. Degree in Geology. I have been involved in the fields of exploration, mine geology and resource estimation for a wide range of mineral commodities in North America, Europe, South America, Africa and Asia and practiced my profession continuously since 1986;
3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of Ontario, registration number (APGO#1235);
4) I have personally visited the Roughrider West and East Zones and project area from September 13 and 14, 2010 and March 16 to March18, 2011 and ;
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the QP responsible for Sections 3 to 11, 13, and 22 of this technical report;
8) SRK Consulting (Canada) Inc. was retained by Hathor Exploration Limited to prepare a technical report for Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan in accordance with National Instrument 43-101 and Form 43-101F1 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Hathor Exploration Limited personnel;
9) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Roughrider Project or securities of Hathor Exploration Limited;
10) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
11) I consent to the filing of the technical report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public of extracts from the technical report; and

PROFESSIONAL GEOSCIENTIST
G. DAVID KELLER
PRACTISING MEMBER
1235
ONTARIO

Toronto Canada
October 26, 2011

G. David Keller, P.Geo.
Principal Consultant (Resource Geology)

U.S. Offices:
Anchorage 907.677.3520
Denver 303.985.1333
Elko 775.753.4151
Fort Collins 970.407.8302
Reno 775.828.6800
Tucson 520.544.3688

Mexico Office:
Guadalupe, Zacatecas
52.492.927.8982

Canadian Offices:
Saskatoon 306.955.4778
Sudbury 705.682.3270
Toronto 416.601.1445
Vancouver 604.681.4196
Yellowknife 867.873.8670

Group Offices:
Africa
Asia
Australia
Europe
North America
South America

srk consulting

SRK Consulting (Canada) Inc.
Suite 205 - 2100 Airport Drive
Saskatoon, SK S7L 6M6

T: +1.306.955.4799
F: +1.306.955.4750

Saskatoon@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Mark Liskowich, P.Geo

I, Mark Liskowich, am a Professional Geologist, employed as a Principal Consultant - Environment with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan. I graduated with a BSc (Geology) degree from the University of Regina in Regina, Saskatchewan in May 1989.

I have practised my profession within the mineral exploration and mining industry since 1989. I have been directly involved, professionally in the environmental and social management of mineral exploration and mining projects covering a wide range of commodities since 1992 with both the public and private sector. My areas of expertise are environmental management, environmental auditing, project permitting, licensing, public and regulatory consultation.

As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I did not visit the Roughrider project site, however I have extensive knowledge of the area based on numerous visits to the region.

I am the responsible QP for Section 19 of the "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am independent of Hathor Exploration Ltd. as independence is described by Section 1.5 of NI 43-101.

I have not previously been involved with the Roughrider Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL GEOSCIENTIST
M. LISKOWICH
SASKATCHEWAN
YR. MN. DAY
11/10/26

Mark Liskowich, P.Geo

Dated: October 26, 2011

U.S. Offices:		Mexico Office:	Canadian Offices:		Group Offices:
Anchorage	907.677.3520	Guadalupe,	Saskatoon	306.955.4778	Africa
Denver	303.985.1333	Zacatecas	Sudbury	705.682.3270	Asia
Elko	775.753.4151	52.492.927.8982	Toronto	416.601.1445	Australia
Fort Collins	970.407.8302		Vancouver	604.681.4196	Europe
Reno	775.828.6800		Yellowknife	867.873.8670	North America
Tucson	520.544.3688				South America

srk consulting

SRK Consulting (Canada) Inc.
Suite 2200 - 1066 West Hastings Street
Vancouver, BC V6E 3X2

T: +1.604.681.4196
F: +1.604.687.5532

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Bruce Murphy, FSAIMM

I, Bruce Murphy, a Fellow of the South African Institute of Mining and Metallurgy, am employed as a Principal Consultant – Rock Mechanics with SRK Consulting (Canada) Inc

This certificate applies to the technical report titled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am a Fellow of the South African Institute of mining and Metallurgy. I graduated with a MSc.Eng (Mining) degree from the University Witwatersrand, in May 1996.

I have been involved in mining since 1990 and have practised my profession continuously since then. I have been involved in mining operations, mining related rock mechanics and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Roughrider Project site in June 2011.

I am the responsible QP for Section 23 of the "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am independent of Hathor Exploration Ltd. as independence is described by Section 1.5 of NI 43-101.

I have not previously been involved with the Roughrider Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Bruce Murphy, FSAIMM.

Dated: October 26, 2011

SRK Consulting (Canada) Inc
Suite 2200 - 1066 West Hastings Street
Vancouver, BC V6E 3X2

T: +1.604.681.4196
F: +1.604.687.5532

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Cameron C. Scott, P.Eng.

I, Cameron C. Scott, am a Professional Engineer, employed as a Principal Consultant with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I have been a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11523) since 1978. I graduated with a B.A.Sc. Degree in Geological Engineering from the University of British Columbia in 1974 and subsequently was granted an M.Eng. Degree in Civil Engineering (Geotechnical Option) by the University of Alberta in 1984.

I have worked as a Geotechnical Engineer for a total of 37 years. Most of my professional practice has focused on the geotechnical and hydrogeological aspects of mining, including the site selection, design, permitting, operation and closure of mine waste facilities in Canada, the US, Mexico, Central and South America, Europe and various countries within the former Soviet Union.

As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Roughrider Project site on June 27, 2011.

I am the responsible QP for Section 17.1 of the "Preliminary Assessment Technical Report for the East and West Deposits Roughrider Uranium Project, Saskatchewan", effective date of September 13, 2011.

I am independent of Hathor Exploration Ltd. as independence is described by Section 1.5 of NI 43-101.

I have not previously been involved with the Roughrider Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Cameron C. Scott, P.Eng.

Dated: October 26, 2011